A STUDY OF

SHOTFIRING IN MINES

WITH SPECIAL REFERENCE TO SAFETY

BY

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CHAPTER I

INTRODUCTION

Although explosives have their origin in Chinese fireworks, known to have been in existence for over 2,000 years, their useful properties were not exploited in the winning of minerals underground until the beginning of the seventeenth century. The earliest reference to their use occurs in the proceedings of the Schemnitz Mine Tribunal for 1627, which include a report of a demonstration of blasting given in a Hungarian mine. However, such was the convenience of even the early types of explosive that their use spread rapidly, and by 1689 gunpowder was in use in Cornish tin mines.

All the early types of explosives were variations of gunpowder with three essential constituents - charcoal, sulphur and potassium chlorate - providing respectively the fuel, the means of easy ignition and the oxygen required for the combustion of the charcoal and the sulphur. Although no records exist, it is certain that many accidents would be caused both by the primitive nature of the explosives and carelessness on the part of the men engaged on shotfiring operations.

Immediately, therefore, in the study of shotfiring accidents a differentiation must be made between those caused by failure of the human element and those caused by the failure of the explosive or ancillary equipment to attain technical perfection. The Chief Inspector of Explosives in/

in his annual report sub-divides shotfiring accidents occurring in mines and quarries into sixteen categories:

Prematures and failure to get away from the shothole
Firing while persons are at shothole
Projected debris
Returning too soon, hang fires
Tampering with misfires
Ramming and stemming
Sparks, flames etc.
Boring into unexploded charges
Striking unexploded charges in removing debris
Detonator accidents (a) Preparing charges
(b) Detonator accidents other than above
Lighting fuse before inserting charge
Fumes
Hot or unexploded residue left in socket
Various (i)

Details of ignitions of gas following shot-:firing and accidents resulting from such ignitions are given in a separate section of the report.

Before differentiating this classification into two main categories, it is first essential to decide the requirements of a perfect explosive, detonator and exploder. An explosive will ideally

(a) Be incapable of igniting methane or coal dust under any conditions.

(b) Always detonate completely when correctly initiated and never under any other circumstances.

(c) Not produce toxic fumes.

Similarly a detonator should

(a)/

(a) Always fire on the application of the correct firing current, or when the fuse has burned down and in no other circumstances.

(b) Be incapable of igniting methane or coal dust under any conditions,

and an exploder should

(a) Always fire the detonator or detonators when correctly used.

(b) Be incapable of igniting methane or coal dust.

These requirements are not at present all attained.

Considering the division further, it is clear that in the category containing accidents caused by the failure of the explosive and other equipment to attain perfection we must include:

(1) Accidents resulting from ignitions of methane and/or coal dust by explosives, detonators or exploders.

(2) Prematures

(3) Ramming and stemming

(4) Sparks, flames etc.

(5) Boring into unexploded charges

(6) Striking unexploded charges in removing debris

(7) Detonator accidents

(8) Fumes

(9) Hot or unexploded residue left in socket.

Similarly, accidents caused by errors of judgement will include the following sub-divisions:

(1) Firing while persons are at hole

(2) Failure to take proper cover

(3) Projected debris

(4) Returning too soon.

Accidents caused by tampering with misfires are included in the second category as the major contributing/

TABLE I

Numbers Killed and Injured by Accidents Resulting from the Use and handling of Explosives in Mines and Quarries 1932 - 1956

Year	A: Accidents Caused by Failure of Human Element					B: Accidents Caused by Failure of Explosive and Ancillary Equipment to attain Perfection									C	Sub Totals			Total	Totals expressed as percentages A+B+C = 100%		
	Not taking Proper Cover	Firing when Persons are at Hole	Projected Debris	Tampering with Misfires	Returning too soon	Ignitions of Methane	Prematures	Ramming and Stemming	Sparks Flames	Boring into Unexploded Charges	Striking Charges in Debris	Detonator Accidents	Fumes	Unexploded Residue left in socket	Unclassified	A	B	C	A+B+C	A	B	C
1932	105	10	2	6	28	18	10	6	32	10	1	8	7	2	26	151	94	26	271	56	35	9
33	97	3	5	4	12	7	17	4	23	12	0	14	4	7	18	121	88	18	227	53	39	8
34	94	12	6	2	24	33	17	8	27	10	10	15	6	5	8	138	131	8	277	50	47	3
1935	130	17	1	13	23	1	13	10	25	27	5	8	5	1	10	184	95	10	289	64	33	3
36	113	20	3	1	15	2	18	11	21	18	6	11	2	1	18	152	90	18	260	58	35	7
37	138	10	1	7	35	0	20	6	26	11	4	15	6	2	19	191	90	19	300	64	30	6
38	152	22	0	5	24	9	13	12	22	22	7	7	9	5	25	203	106	25	334	61	32	7
39	148	22	1	6	30	78	18	11	18	11	10	11	3	2	22	207	162	22	391	53	41	6
1940	124	23	2	1	34	0	14	8	18	17	6	11	5	2	12	184	81	12	277	66	29	5
41	169	10	3	2	27	2	10	2	12	10	1	13	8	4	11	211	62	11	284	74	22	4
42	182	11	1	1	33	15	12	9	4	11	3	21	3	1	12	228	79	12	319	71	25	4
43	176	18	0	1	23	8	20	6	12	4	2	19	2	1	4	218	74	4	296	74	25	1
44	215	15	3	0	24	32	15	1	3	3	0	12	3	0	12	257	69	12	338	76	20	4
1945	199	12	3	2	15	9	10	0	6	5	1	11	2	2	6	231	46	6	283	82	16	2
46	214	23	1	4	23	4	9	2	6	8	2	16	3	3	7	265	53	7	325	82	16	2
47	244	21	1	1	27	130	4	5	6	5	1	12	4	1	2	294	168	2	464	63	36	1
48	274	25	8	7	19	0	11	8	0	13	0	18	4	1	13	333	55	13	401	83	14	3
49	266	21	20	3	28	0	39	3	0	4	0	15	2	1	11	338	64	11	413	82	15	3
1950	241	19	26	2	10	5	12	2	2	5	1	10	9	2	14	299	48	14	361	83	13	4
51	214	13	34	2	12	4	6	2	0	11	3	8	0	1	8	275	35	8	318	85	11	4
52	246	36	17	4	8	0	14	3	2	7	3	9	1	0	13	311	39	13	363	86	11	3
53	248	36	14	2	4	8	8	4	0	7	4	11	0	5	10	304	47	10	361	84	13	3
54	222	27	5	0	1	0	13	0	0	12	0	6	0	2	11	255	33	11	299	86	10	4
1955	220	27	13	0	2	0	7	5	0	3	7	7	0	0	5	262	29	5	296	88	10	2
56	228	23	1	2	0	4	18	2	0	4	0	7	0	0	0	254	35	0	289	88	12	0

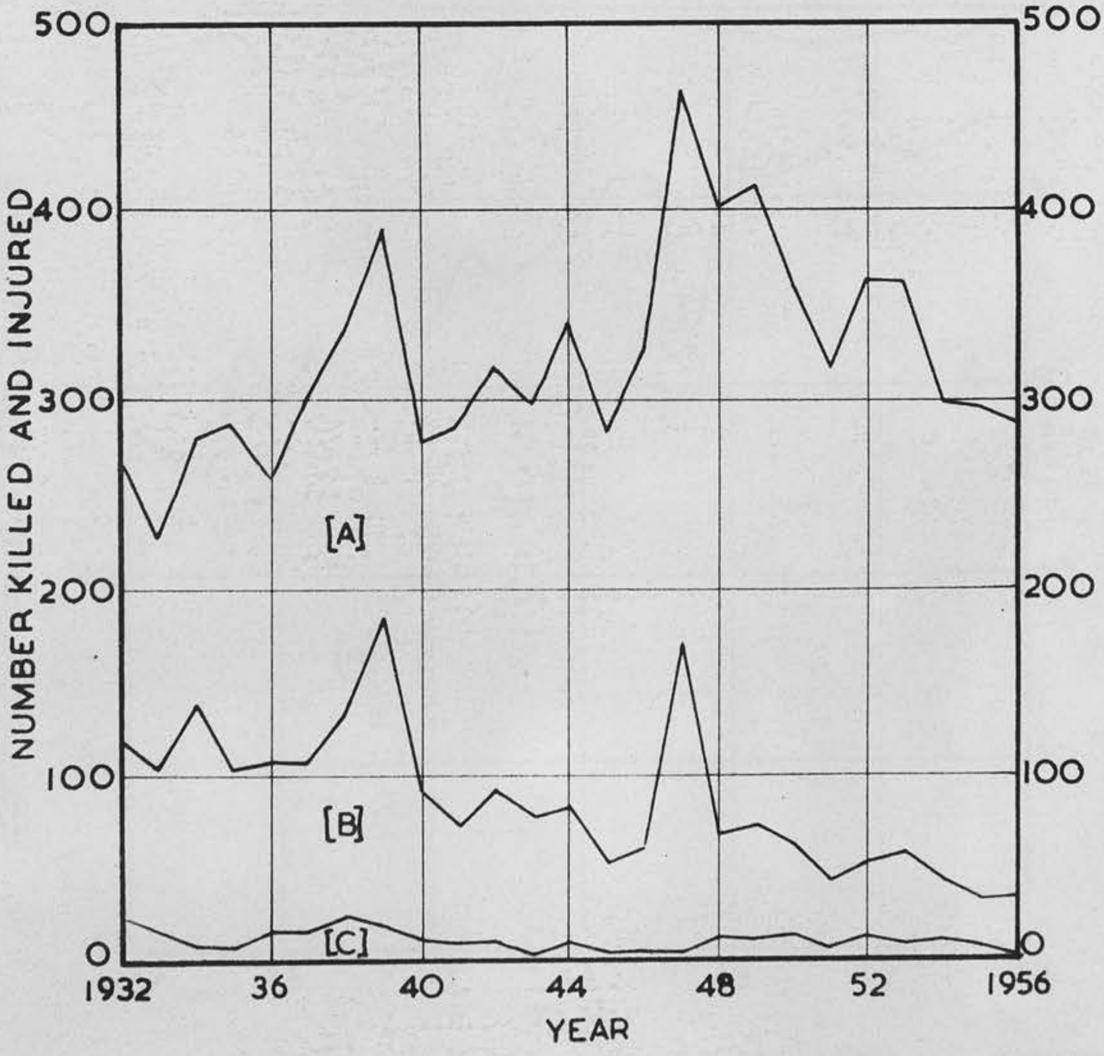
GRAPH I

NUMBERS KILLED AND INJURED IN USE AND HANDLING OF EXPLOSIVES IN MINES AND QUARRIES [1932-1956] IN ACCIDENTS:-

[A] CAUSED BY FAILURE OF HUMAN ELEMENT

[B] CAUSED BY FAILURE OF EXPLOSIVE OR ANCILLARY EQUIPMENT TO ATTAIN PERFECTION

[C] NOT STATED OR VARIOUS



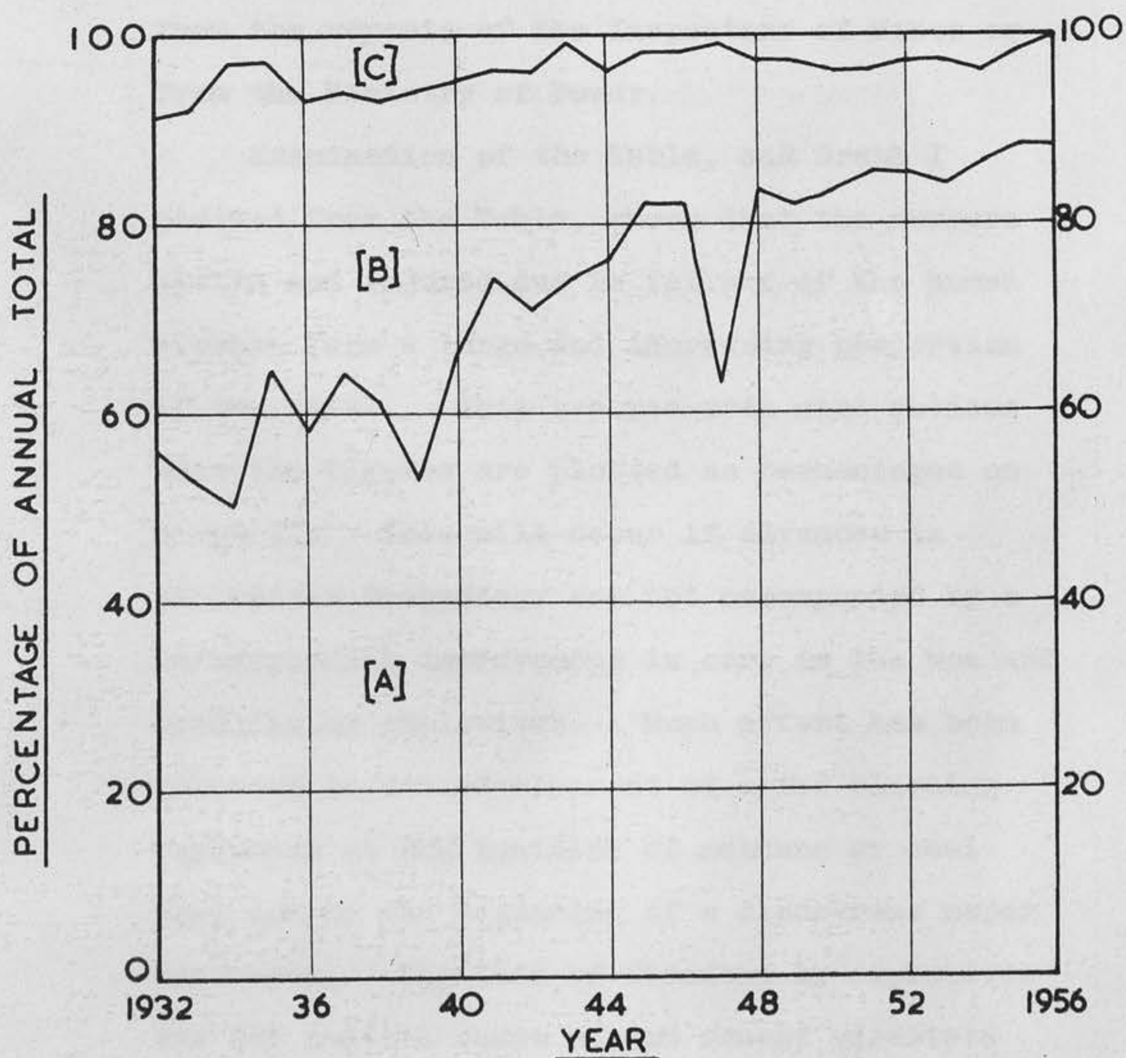
GRAPH II

NUMBERS, EXPRESSED AS PERCENTAGES OF THE TOTAL, KILLED AND INJURED IN THE USE AND HANDLING OF EXPLOSIVES IN MINES AND QUARRIES [1932-1956] IN ACCIDENTS:-

[A] CAUSED BY FAILURE OF THE HUMAN ELEMENT

[B] CAUSED BY FAILURE OF EXPLOSIVE OR ANCILLARY EQUIPMENT TO ATTAIN PERFECTION

[C] NOT STATED OR VARIOUS



contributing factor is gross carelessness on the part of the men engaged on the job. Accidents under the heading 'Lighting fuse before inserting charge' are included in the 'unclassified' group, as this practice has resulted in a negligible number of injuries over the period 1932 - 1956.

In Table I the figures recorded refer to the number of people killed and injured in mines and quarries. Unfortunately a similar sub-divison is not available for mines alone, either from the reports of the Inspectors of Mines or from the Ministry of Power.

Examination of the Table, and Graph I plotted from the Table, shows that the numbers killed and injured due to failure of the human element form a large and increasing proportion of the total. This becomes even more obvious when the figures are plotted as percentages on Graph II. This will occur if advances in explosives technology are not accompanied by a corresponding improvement in care in the use and handling of explosives. Much effort has been expended in the development of safer blasting equipment as any ignition of methane or coal dust can be the beginning of a disastrous major explosion. Ignition of firedamp by explosives was the initial cause of two recent disasters at Valleyfield Colliery, Fife in 1939 and the William Pit, Whitehaven in 1947. In addition, the/

the subject is amenable to laboratory investigations under controlled conditions, enabling the results obtained to be easily analysed.

On the other hand, accidents caused by errors of judgement or carelessness seldom kill or injure more than one or two people at a time, and are not at all amenable to laboratory investigations. Nevertheless, as this class now accounts for approximately 86% of the annual total, there seems good reason for revision of thought on the problem. Over the years 1933-57, only one paper published in the Transactions of the Institution of Mining Engineers dealt with the problem of shotfiring accidents caused by failure of the human element, compared with nineteen which brought advances in blasting technology to the notice of the Institute members (ii, iii). The main part of this work is therefore concentrated on examining shotfiring practices and methods, and determining the reasons for the carelessness and weaknesses in the methods employed which result in shotfiring accidents. It is, however, considered essential to review the advances made in the field of explosives technology.

R E F E R E N C E S/

REFERENCES

(i) Annual Reports of H. M. Chief Inspector of Explosives

(ii) Transactions of the Institution of Mining Engineers, LXXXVI - CXVI

(iii) do. CXIII, 926 - 947

CHAPTER II

DEVELOPMENT OF MODERN MINING EXPLOSIVES

All the early explosives were derivatives of gunpowder and were thus non-detonating and comparatively slow burning. Although satisfactory for metal mining work, such explosives were very dangerous in the presence of inflammable or explosive atmospheres, as any ignition which occurred could give rise to a disastrous coal dust explosion which could sweep through the entire colliery workings.

Although gunpowder and its derivatives were the only commercial explosives available up to the middle of the 19th century, other substances had been discovered which were to assume an increasingly important role in blasting work. Howard in 1799 produced fulminate of mercury and this was used as a detonating agent in firearms by 1814. Pelouze in 1838 initiated the study of nitrocelluloses by treating cotton with strong nitric acid. Schonbein and Sobrero employed the dehydrating and nitrating action of sulphuric and nitric acids to produce guncotton and nitroglycerine from cotton and glycerine. These substances, requiring a sudden shock or blow to make them explode, opened out new possibilities for blasting work. However, little notice was taken of them until 1859 when Alfred Nobel and his father began a serious study of their manufacture, and not until 1863 was nitroglycerine prepared and used commercially. Pure nitro-:glycerine is an oily liquid freezing at 13°C. and/

and is, especially in the frozen state, extremely sensitive to shock of any kind. It is thus unsuited for use in mines and to minimise the inherent danger Nobel experimented and found that a mixture of 75% nitroglycerine and 25% kieselguhr was comparatively safe and easy to handle or form into cartridges. This mixture, Dynamite, soon became an established blasting agent and is still used in certain metalliferous mines abroad. Because of the high toxicity of the fumes produced on detonation it is not, however, suitable for general use underground.

In 1875 Nobel discovered that the mixture of 8% of guncotton and 92% nitroglycerine was at once easily formed into cartridges and waterproof. This explosive, Blasting Gelatine, is still the most powerful commercially available and is used as a standard by which to measure the strengths of coal mining and other explosives. To reduce the strength of this explosive for certain applications, Nobel lowered the nitro:glycerine content, added woodmeal and corrected the resultant oxygen deficiency by adding potassium nitrate. This mixture was called Gelatine Dynamite. In 1879 Nobel took out a patent for a similar mixture containing ammonium nitrate in place of the potassium nitrate and thus introduced a new extremely important class of explosives - the ammonium nitrate/

nitrate derivatives.

Nitroglycerine and explosives containing nitroglycerine are extremely sensitive to shock in the frozen state and many accidents were caused by people handling or attempting to thaw out explosives. Although a patent was taken out in 1866 for a method of lowering the freezing point of nitroglycerine, low freezing nitroglycerine preparations did not appear on the British Permitted List until 1925 and were not made compulsory till 1933. The most suitable substance for lowering the freezing point of nitroglycerine is nitroglycol which exhibits comparable explosive qualities but freezes at -22°C. When 20% of glycol and 80% of glycerine are mixed and the mixture nitrated, the resultant combination will not freeze at any winter temperatures experienced in this country.

Although other compounds, generally the products of nitration of organic substances such as benzene and toluene, have been used in mining explosives they have never achieved importance comparable with nitroglycerine and ammonium nitrate and their derivatives

Early Experimental work on the ignition of Methane and Coal Dust by Explosives

The dangers of coal dust and the influence of coal dust on mine explosions was realised early in this country and the views of Bald (i) were supported by Buddle, who investigated the Wallsend/

Wallsend explosion in 1830 (ii). It was not universally held that coal dust was an important factor and the Coal Mines Regulation Act of 1872, containing regulations governing the use of 'gunpowder or other explosive or inflammable substance' underground, did not detail any precautions to be taken against coal dust.

An early attempt to ensure safety with the dangerous explosives of the day was the development of the water cartridge, patented in 1876 by Captain McNab. This consisted of a paper bag filled with water in which the explosive was detonated, and although tin containers later provided a more reliable and stronger vehicle for the water, the inconvenience in their use resulted in a comparatively early demise in spite of the undoubted safety advantages gained.

In 1875, Galloway proved conclusively that coal dust alone or with methane could produce violent explosions when initiated by an intense igniting source. As a result of several disastrous explosions in this country and abroad, Commissions were set up to examine the problem and as explosives were an obvious and ready means of ignition it became clear that shotfiring must be subjected to a searching examination. The first of these Commissions - Le Commission du Grisou - was set up in France in 1877 to determine if by modifying the nature, method/

method and use of explosives, explosions from this cause could be diminished. The committee concluded in 1880 that, as the ignition of methane occurred between 600°C. and 700°C., and no explosive existed which exploded at a lower temperature, it would be difficult to prevent entirely the ignition of methane by shotfiring.

In Britain a Royal Commission on Accidents in Mines was appointed in 1879 and reported in 1886 that the following facts were conclusively established.

(1) "The occurrence of a blown-out shot in working places where inflammable dust exists in great abundance may, even in the total absence of firedamp, give rise to violent explosions or at any rate be followed by the propagation of flame through considerable areas.

(2) "The occurrence of a blown-out shot where only small percentages of firedamp exist in the presence of comparatively slightly inflammable or even non-inflammable but very fine dry dusts may give rise to an explosion."

As a result of experiments with various types of explosives the Commission concluded that:

(1) "The employment of nitroglycerine or guncotton preparations in conjunction with the water cartridge secures safety against explosions even by a blown-out shot where the air may contain a highly inflammable dust

suspended/

suspended when the shot is fired.

(2) "The employment of the above types of explosive in conjunction with porous tamping soaked with water ensures safety in the above circumstances and also if the shot blows out into a highly inflammable mixture of firedamp and air.

(3) "Neither the water cartridge nor porous tamping afford protection against explosions of coal dust or firedamp when employed with powder explosives." (iii)

<u>The Development of the British Gallery Test</u>

The conclusions of the Royal Commission on Accidents in Mines were embodied in the Coal Mines Regulation Act of 1887 which stated that where gas was present or where the place was dry and dusty no shot should be fired "unless the explosive . . . is so used with water or other contrivance as to prevent it enflaming gas, or is of such a nature that it cannot enflame gas". When this Act came into force in January, 1888, many so-called flameless explosives were manufactured, and as tests by makers and users generally gave contradictory results, those responsible for safety in mines were left in a state of doubt. Accordingly, a committee was appointed by the North of England Institute of Mining and Mechanical Engineers to investigate and report on the safety of explosives. The test/

test employed was designed to simulate the conditions of a blown-out shot, with the explosive doing no work and firing into an inflammable and explosive atmosphere. The gallery was 3 feet in diameter and 101 feet long, and the explosive, contained in a cannon 42½ inches long with a bore of 1⅛ inches, fired into an explosion chamber 22½ feet long formed off by a paper seal from the rest of the gallery. A series of tests established that high explosives, although producing evident flame on detonation, were much less dangerous and less liable to ignite inflammable mixtures of air and methane and/or coal dust than blackpowder.

Parallel work, carried out at the same time for a Royal Commission on Explosions from Coal Dust, confirmed these results and the Coal Mines Regulation Act of 1896 conferred on the Secretary of State powers to propose, modify and amend rules on such subjects as the use and storage of explosives at mines. A special provision (Section 5) empowered him to prohibit by order the use of any explosive likely to be or become dangerous in any mine, and, by virtue of these powers a series of Explosives in Coal Mines Orders was issued prescribing the nature and manner of use of explosives to be employed in coal mines.

To provide a basis for differentiating between explosives a committee (iv) appointed in 1896/

1896 " to enquire into the best tests to determine the safety of explosives . . . and the means of applying such tests", recommended the erection of a testing station. This suggestion was adopted and a station opened for official work at Woolwich in June 1897. The first official gallery differed only in dimensions from that erected at Hebburn by the North of England Institute of Mining and Mechanical Engineers. The test consisted of firing 40 shots of explosive, each well stemmed with dry clay and equivalent to 2 oz. of 75% dynamite or 6 oz. of R.F.G. 2 gunpowder for high explosives or powders respectively, into a 10% coal gas/air mixture. If, after 20, 30 and 40 shots had been fired, and no ignition or not more than one or two ignitions respectively had occurred, the explosive was considered to have passed. In 1899 a more stringent supplementary test with heavier charges and a 15% coal gas/air mixture was introduced and in 1901 the original permitted list was withdrawn and only those explosives passing the more severe test were allowed in mines governed by the Explosives in Coal Mines Orders.

In 1907 a Departmental Committee on Bobbinite concluded that the existing standard was not sufficiently high, and this report was endorsed by a Royal Commission on Mines in its second report dated July 1909. A new experimental station/

station was opened at Rotherham in 1911 and, after initial work on explosives already on the Permitted List, official testing was started in July 1911. By March 1914 only those explosives passing the Rotherham test were allowed on the new Permitted List, The new gallery was 50 feet long, 5 feet in diameter with an explosion chamber 18 feet long, and the cannon had a bore of 2.17 inches and was 47 inches long. The test consisted of finding the maximum charge which when fired untamped into a coal gas/air atmosphere did not cause ignition to five successive shots. This charge was also tested with coal dust/air mixtures and again no ignitions in five successive shots were required.

In 1921, because of difficulties in obtaining standard test conditions due to variations in coal gas composition, testing was transferred to an identical station at Ardeer, erected for experimental work by the Nobel Division of Imperial Chemical Industries. The test remained unchanged but work carried out at Ardeer, Eskmeals, Rotherham and Buxton culminated in the introduction of a modified test in 1929, coinciding with the transference of official testing to Buxton. The new test, which has remained unchanged for unsheathed permitted explosives, consisted of:

(1) Firing into a 9% methane/air mixture

(a) Five shots of 8 oz. unstemmed

(b)/

(b) Five shots of 28 oz. stemmed with one clay plug.

(2) Firing into a coal dust suspension

(a) Five shots of 28 oz. stemmed with one clay plug.

For explosives with a density of less than 0.75 gms./cc., the charge used in the stemmed shots is reduced to 18 oz. All shots are directly initiated, i.e. with detonator nearest the muzzle of the cannon.

The Development of Sheathed and Equivalent to Sheathed Explosives

Manufacturers were well able to produce explosives satisfying official requirements by reducing the nitroglycerine content and adding inert substances. Abroad work was concentrated on the development of even safer explosives, culminating in the successful development and production of sheathed explosives in 1913. Although the discovery is popularly attributed to Lemaire, Jikinski in 1888 had used sand or kieselguhr saturated with water as a sheathing material and in 1898 an explosive 'Elephant Brand Gunpowder' was authorised by the Home Office provided that the cartridge was surrounded by a layer of sodium bicarbonate and separated from it by a suitable diaphragm. It was however due to the work of Watteyne and Lemaire that sheathed explosives became commercially practicable. It is interesting to note/

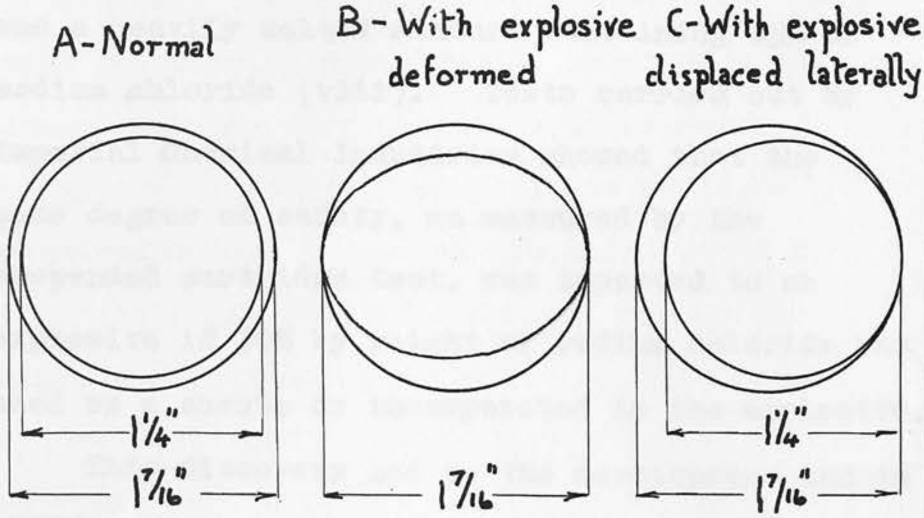
note that they arrived at the final solution after research with external tamping - incom-:bustible dust placed in front of the shothole - so placed that the blast raised a cloud of particles inhibiting ignition of gas or coal dust. These experiments were so successful that they led logically to the attachment of the incombustible material to the explosive in a tube around the charge. After experimenting with many substances Watteyne and Lemairefinally chose a mixture of sodium chloride and calcium fluoride as offering the best compromise between cost and efficiency. In 1920 Belgian regulations permitted sheathed explosives as an alternative to external tamping subject to certain conditions regarding diameters of charge and sheath, and methods of manufacture. (v)

Introduction of sheathed explosives to Britain was delayed until 1934 when Naylor and Wheeler demonstrated the greatly enhanced safety obtained by their use. The explosive used in their tests, carried out for the Explosives in Mines Research Committee of the Safety in Mines Research Board, had a composition of nitroglycerine 12.5%, ammonium nitrate 75% and woodmeal 12.5%, and the smallest weights causing ignition when fired freely suspended in a 9% methane/air mixture were found. Unsheathed, ignitions were caused by charges of less than two ounces, but sheaths of sodium bicarbonate and sodium hyposulphite/

hyposulphite allowed 10 oz. to be fired safely and iron filings allowed 12 oz. to be fired. (vi)

Further tests showed that sodium bicarbonate gave similar protection to all permitted explosives and, as manufacturing difficulties were overcome, sheathed explosives were brought into commercial production and remained unchanged until 1949. Then the Nobel Division of I.C.I. Ltd. introduced a felt sheath containing 85% of sodium bicarbonate and 15% cellulose fibre. This was wrapped round the charge and glued in position, overcoming one of the great disadvantages of powder sheaths - liability to damage and consequent reduction in safety factor. Shepherd and Grimshaw (vii) found that one type of permitted explosive, which caused ignition of a 9% methane/air mixture with charges of over 2 oz. fired freely suspended, had the limiting weight increased to 8 oz. when uniformly sheathed with 3/32 inch of sodium bicarbonate.

FIGURE I - SHEATHED EXPLOSIVE



If/

If however the explosive cartridge was deformed into an ellipse with a major axis 1 7/16 in. diameter (Figure I) or displaced laterally so that the sheathing was nil on one side and 3/16 in. on the other, the limiting charge was reduced to 4 oz. Felt sheathing, although largely removing the possibility of accidental damage, does not preclude intentional removal of the sheath by uninformed workmen.

It was felt that if the sheathing material could be incorporated in the explosive enhanced safety would result. Sodium bicarbonate is not, however, suitable for incorporation into ammonium nitrate explosives as these two compounds react with the evolution of ammonia. Common salt, sodium chloride, can however be included and is of comparable efficiency with sodium bicarbonate when used as a sheath. Due to its hygroscopic nature it cannot be used in this way. As early as 1910 work was carried out in Belgium to determine the influence of the size of the gallery on testing and one of the explosives used was a heavily salted mixture containing 25% of sodium chloride (viii). Tests carried out by Imperial Chemical Industries showed that the same degree of safety, as measured by the suspended cartridge test, was imparted to an explosive if 30% by weight of sodium chloride was used as a sheath or incorporated in the explosive.

This discovery led to the development, and in 1950/

1950, the marketing of a new range of explosives designated Equivalent to Sheathed, generally contracted to Eq. S. To aid the attainment of a minimum standard for Sheathed and Eq. S. explosives a new test was formulated and came into effect in 1953. This consisted of:

(1) Firing into a 9% methane/air mixture

(a) 5 shots of 20 oz., including weight of sheath, inversely initiated, without stemming.

(b) 5 shots of 28 oz., excluding and without sheath, directly initiated, with 1 in. clay plug.

(2) Firing into a coal dust/air mixture

(a) 5 shots of 20 oz., including weight of sheath, inversely initiated.

For explosives with a density less than 0.75 gms./c.c., the charge in test (1) (b) is reduced to 18 oz.

Other Testing Procedures

The first test at Hebburn, and the present official testing procedure, were both designed to simulate the condition of a shot doing no work and blowing out directly into the most dangerous atmosphere possible underground. As a deplorable standard of practice would be required before 28 oz. of explosive would be fired with one inch of stemming, or any shot without any stemming whatever, it soom became evident that the official test was not in fact reproducing/

reproducing the conditions under which ignitions were occurring in the field. Following the introduction to this country of sheathed explosives in 1933, Naylor and Wheeler developed the suspended cartridge test (ix) in order to have available a sensitive criterion for the differentiation of permitted explosives, some of which gave ignitions with charges of only 2 oz. It seemed unlikely that such conditions could occur in practice but Grimshaw (x) after investigating many cases of ignition of methane following shotfiring, in the main when sheathed explosives were being used, concluded that in the majority of cases part of the explosive charge had been crossed by a break or fissure in the strata and would therefore be in direct contact with the atmosphere in the break.

Experimenting with charges partially confined in a very short hole he found that any given charge was more likely to produce ignitions under these conditions than when fired wholly confined or wholly unconfined. Following up this work by firing shots exposed to an artificial break, he found that such shots could cause ignitions with charges much below the legal maximum, and a test was designed to enable the safety of explosives under these conditions to be determined under controlled laboratory conditions. This consists (xi) of firing charges from a cannon facing directly into the space/

space between two parallel steel plates. At present this is purely a research apparatus and forms no part of the official test.

Development in explosives technology abroad has followed a roughly parallel trend with that in this country, with gallery tests imposing minimum standards to be attained by explosives for use in specific circumstances. A notable exception is that of France where, following the investigations of the French Commission in 1887-88 (xii), regulations were introduced specifying maximum calculated detonation temperatures to be allowed for explosives for use in coal and rock, without imposing any charge limits. At the present day a gallery test is employed to ensure that explosives satisfying theoretical requirements do not behave dangerously under practical conditions, and charge limits are imposed on all types of explosive authorised for use in gassy and dusty mines.

An interesting development in the field of explosives testing is the angle mortar test. In this the charge is fired lying in a section cut out of a cylindrical mortar, so placed that the products of detonation impinge on a ricochet plate. The angle of the mortar groove, the distance to the plate and the angle of incidence can all be varied and this test forms part of the official examination for sheathed explosives in/

in Belgium.

Exploders for use in Coal Mines

All exploders which are to be used in a mine must be approved by the Ministry of Power, and in a mine or part of a mine where permitted explosives are required to be used exploders must be additionally approved as safe for use in gassy atmospheres. (xiii)

The basic essential of all electric exploders is that they should, when correctly operated, be capable of passing adequate current through the external circuit of cable and detonators to ensure that all the detonators fire. With single shot firing, no difficulty is experienced in satisfying this requirement and also making the exploder safe for operation in methane/air mixtures. In operation an armature is rotated between the poles of a set of permanent magnets by a detachable firing key.

Multi-shot exploders approved for use in gassy coal mines are rated at 6-shot capacity, and the principle involved is different from single-shot machines. A $67\frac{1}{2}$ volt dry battery is used to charge a 150 microfarad condenser which is subsequently discharged into the firing circuit. This type incorporates a circuit continuity test powered by a $1\frac{1}{2}$ volt dry battery and operating an ohmmeter or bulb. The test circuit must be disconnected by means of a push button/

button before firing is possible.

For rounds of over six shots no exploder is as yet commercially available which is incapable of igniting firedamp. However, for firing large rounds in drifts and shafts the Divisional Inspector of Mines may give conditional approval to certain unapproved exploders, usually the Beethhoven. In operation, a small alternator, driven by an external detachable handle, has its voltage transformed and rectified to charge a 6 microfarad condenser to a minimum of 1200 volts, indicated by a discharge in a small neon lamp. On firing, the condenser is discharged into the external circuit by operation of a button switch.

This exploder is not by any means foolproof in the presence of explosive methane/air atmospheres and the conditions imposed by the Divisional Inspector must be rigidly observed if safety is to be ensured.

Regulations (xiv) govern the maintenance and testing of all exploders and, in general, little trouble is encountered in their use. Research is now concentrated on the development of an approved device for firing rounds of up to 15 shots in gassy mines, and means to make the firing of large rounds safer.

<u>Detonators for use in Coal Mines</u>

The basic requirement of a detonator is to produce/

produce on firing a shock adequate to initiate detonation in an explosive cartridge in which it is embedded. In electric detonators this is achieved by using the heat generated by the current passing through a fine wire to fire a match-head composition which in turn sets off priming and base charges. The above elements are enclosed in a metal tube and protected from moisture by a neoprene plug round which the neck of the tube is crimped. Where fuse firing is employed, the initial heat is obtained from the burning core of the fuse.

Delay detonators are similar to plain electric detonators but the fusehead is designed to produce no appreciable gas pressure, as this could cause the priming and base charges to operate. Additionally a delay element, variable in composition and length, is introduced between the fusehead and the priming charge, to obtain varying intervals between the passing of the firing current and detonation of the explosive.

The ignition of methane by Explosives and Ancillary Equipment

1. Ignition by Explosives

Three main methods by which methane/air mixtures may be ignited by explosives have been suggested - ignition by hot gases, by solid particles and by adiabatic compression - and no proof has yet been forwarded to discredit entirely or/

or establish beyond doubt the influence of any of these methods.

(a) Ignition by hot gases. Following the discovery by Mallard and Le Chatelier that methane ignited on exposure to temperatures of 600 - 700°C., but only after an appreciable time lag which decreased with increasing temperature the so-called 'French doctrine' of 1890 was developed. This required that explosives on detonation should produce no solid particles or combustible gases, and that the maximum calculated temperature of detonation should not exceed 1900°C. for stone work or 1500°C. for coal. Thus any ignitions obtained would almost certainly be due to hot gases, and the above theory tended to discount the idea of a charge limit, as the temperature produced is independent of the amount of explosive used. This is not borne out in gallery tests as any explosive may be made to cause ignitions by increasing the quantity detonated.

This theory does not explain why the addition of certain salts greatly increases the safety margin of explosives under test conditions. Thus tests (xv) have shown that the safety of an explosive as measured by the official British gallery test is dependent on its strength and density. Nor does the theory explain why certain salts are much more effective than others. Common salt, present in most permitted explosives/

explosives, exerts a very marked effect and is commonly referred to as a 'cooling agent'. As it has been shown (xvi) that a concentration of 4 mgs./litre of fine sodium chloride is sufficient to prevent propagation of flame in highly explosive atmospheres, it would appear that the salt acts on the incendivity of the gallery methane/air mixture as well as that of the hot gases produced on detonation.

(b) Ignition by solid particles. Although there is a great deal of evidence showing that hot particles are produced on detonation of explosives, ignition of methane/air mixtures under such conditions has not been demonstrated conclusively. Photographs (xviii) have shown decomposing and undecomposing particles passing through a 9% methane/air mixture without causing ignition, although these particles have been found capable of igniting guncotton.

(c) Ignition by adiabatic compression. All explosive atmospheres may be ignited by any sudden compression raising the temperature over the required ignition temperature of the mixture, and this effect could well be produced by the pressure created when an explosive detonates. This, however, also applies to alternatives to explosives, such as Cardox, as the required pressure is only about 450 lb. per square inch, and as no ignitions have been recorded with those devices underground, it seems doubtful if the close/

close confinement necessary for the rapid rise in pressure could occur in practice.

2. Ignition by Detonators

The possibility of a methane/air mixture being ignited by a detonator is obviously extremely remote, as the energy produced is very small in comparison with that of the explosive cartridge in which it is embedded. No cases of ignition from this cause have been recorded in recent years.

3. Ignition by Exploders

Although the energy produced by exploders is also small compared with explosives, the possibility of ignitions from this cause is not so remote as with detonators. The mode of ignition is incendive sparking, generally in the cable, caused either by defective joints or the residual energy of the exploder being dissipated at the cable ends after the circuit has been broken by detonation of the explosive. It has not yet been found possible to produce commercially an intrinsically safe exploder for rounds of over six shots and two serious explosions have been attributed to sparking when a Beethhoven exploder was being used to fire large rounds in drifts. It is important to note, however, that the ignition occurs, not in the shothole, but outside it where careful testing will reveal the presence of a dangerous atmosphere, and in both the above explosions the Divisional Inspector in his report found/

found that this elementary precaution had been skimped (xvii).

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(xiv) do. 21.

(xv)/

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CHAPTER III

UTILISATION OF EXPLOSIVES

Explosives, offering an easy and convenient method of preparing coal and stone for hand and power loading, have been widely and increasingly employed in this country since their introduction in 1629. Between 1928 and 1954 the number of shots fired in mines under the 1911 Coal Mines Act increased from 50.4 to 99.4 millions, and over the same period the weight of explosive used increased from 25.5 to 51.9 million pounds. Graph III shows this sharp increase and also divides the total weight used into the different types, i.e. non-permitted, permitted, sheathed permitted and equivalent sheathed permitted.

The total consumption, indicated by the bold black line, has risen almost continuously from the economically depressed days of the 1930's until the present day, but the efforts of the largest user, the National Coal Board, to reduce the use of explosives, coupled with a falling off in coal output have stopped this trend.

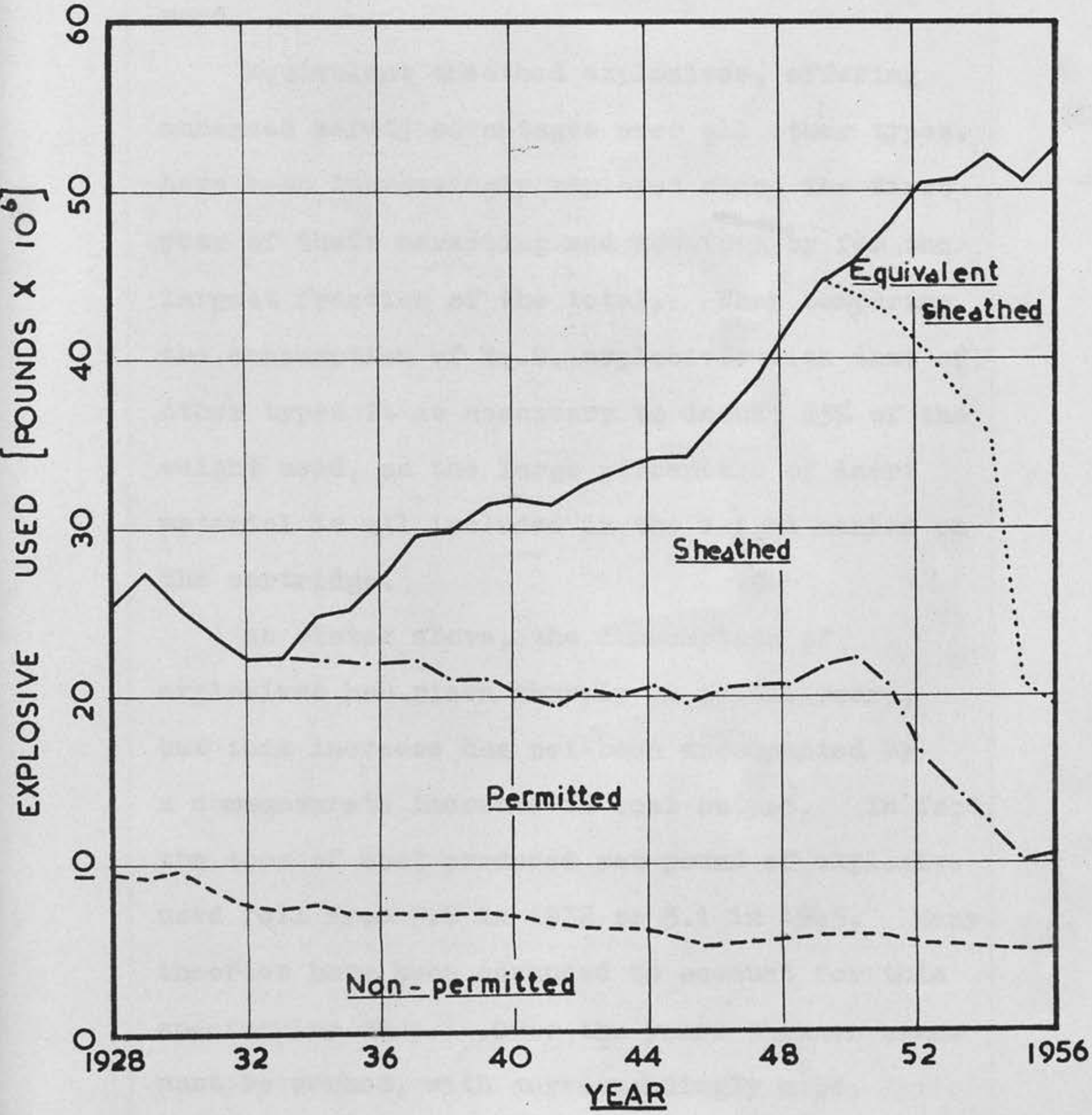
Non-permitted explosives may only be used in naked light mines and shaft sinking operations, and a gradual reduction in their use is evident. Additionally, explosives of this type are generally of very high strength and not entirely suitable for coal getting operations on account of their extreme shattering action, and are now being replaced by weaker types for this work.

Unsheathed permitted explosives were used

to/

GRAPH III

USE OF EXPLOSIVES IN MINES UNDER THE 1911 COAL MINES ACT, OVER THE PERIOD 1928 - 1956



to the extent of 14-16 million pounds per annum between 1922 and 1951, but are now employed mainly in stone mine drivages where the strongest explosive capable of passing the official test is required.

Sheathed explosives formed an increasingly large percentage of the total consumption from the date of their introduction in 1933 until 1949 when equivalent sheathed explosives were introduced. Only a very small quantity is now used.

Equivalent sheathed explosives, offering enhanced safety advantages over all other types, have been increasingly employed since the first year of their marketing and now form by far the largest fraction of the total. When comparing the consumption of Eq.S. explosives with that of other types it is necessary to deduct 25% of the weight used, as the large percentage of inert material is all included in the weight marked on the cartridge.

As stated above, the consumption of explosives has risen sharply in recent years, but this increase has not been accompanied by a commensurate increase in coal output. In fact, the tons of coal produced per pound of explosive used fell from 9.2 in 1932 to 5.1 in 1945. Many theories have been advanced to account for this spectacular drop. Over the years thinner seams must be worked, with correspondingly more, although/

although lighter, shots in coal and more stone work. Again the practice of providing workmen with free explosives instead of requiring each man to buy his own also grew over the same period. Neither of these theories are capable of closer examination, as the necessary figures are not available, but a third, which seeks to relate the increasing use of orthodox longwall working with the fall may be subject to a rough analytical examination.

TABLE II

Relationship of Coal Conveyed and Tons of Coal per Pound of Explosive

% of Coal Conveyed by Machinery y	Tons of Coal per Pound of Explosive x	9.4 - y	log(9.4 - y) Y	logx X
25	9.2	0.2	-0.70	1.38
30	9.1	0.3	-0.52	1.48
37	8.9	0.5	-0.30	1.57
43	8.8	0.6	-0.22	1.63
48	8.6	0.8	-0.09	1.68
51	8.1	1.3	0.11	1.71
54	7.6	1.8	0.26	1.73
58	7.4	2.0	0.30	1.76
61	7.1	2.3	0.36	1.79
64	6.6	2.8	0.45	1.81
65	6.3	3.1	0.50	1.81
66	5.7	3.7	0.57	1.82
69	5.4	4.0	0.60	1.84
71	5.1	4.3	0.63	1.85

When the two quantities, tons of coal per pound of explosive and percentage of coal conveyed by machinery, are plotted for the years over/

over which the most serious decline took place - 1932-45 - it is apparent that a relationship does exist as indicated in Graph IV, although this is certainly not linear. Rather does the Graph appear to be of the form

$$y = ax^b + c$$

where y = tons of coal per pound of explosive

x = percentage of coal conveyed by machinery

b and a constants

c = intercept on the y axis

Extrapolation gives c a value of 9.4.

To determine the values of the constants, the equation is put into the form

$$-y = -c + ax^b \quad \text{hence}$$

$$(c - y) = ax^b \quad \text{and taking logs to the base 10 of both sides}$$

$$\log(c - y) = \log ax^b$$

$$= \log a + b \log x$$

Let $Y = \log(c - y)$

$A = \log a$

$X = \log x$

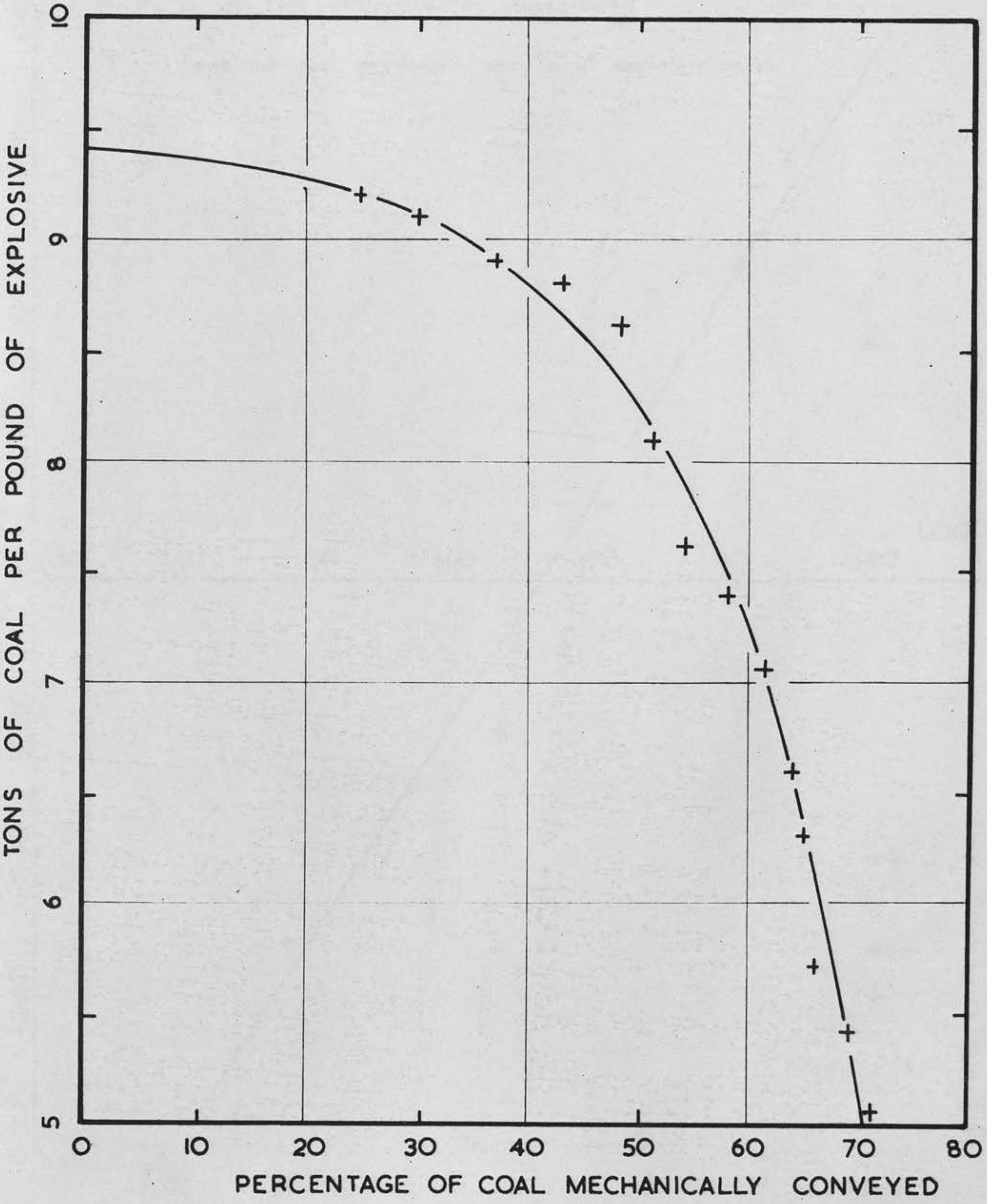
So that

$Y = A + bX$ i.e. the equation of a straight line.

As seen from Graph V, a plot approximating to a straight line is obtained and calculation of the gradient and intercept on the Y-axis gives values for A and b of -5.19 and 3.13. Hence the value of a, taking the antilog of -5.19, is found to be 6.46×10^{-6}, and the equation of the curve/

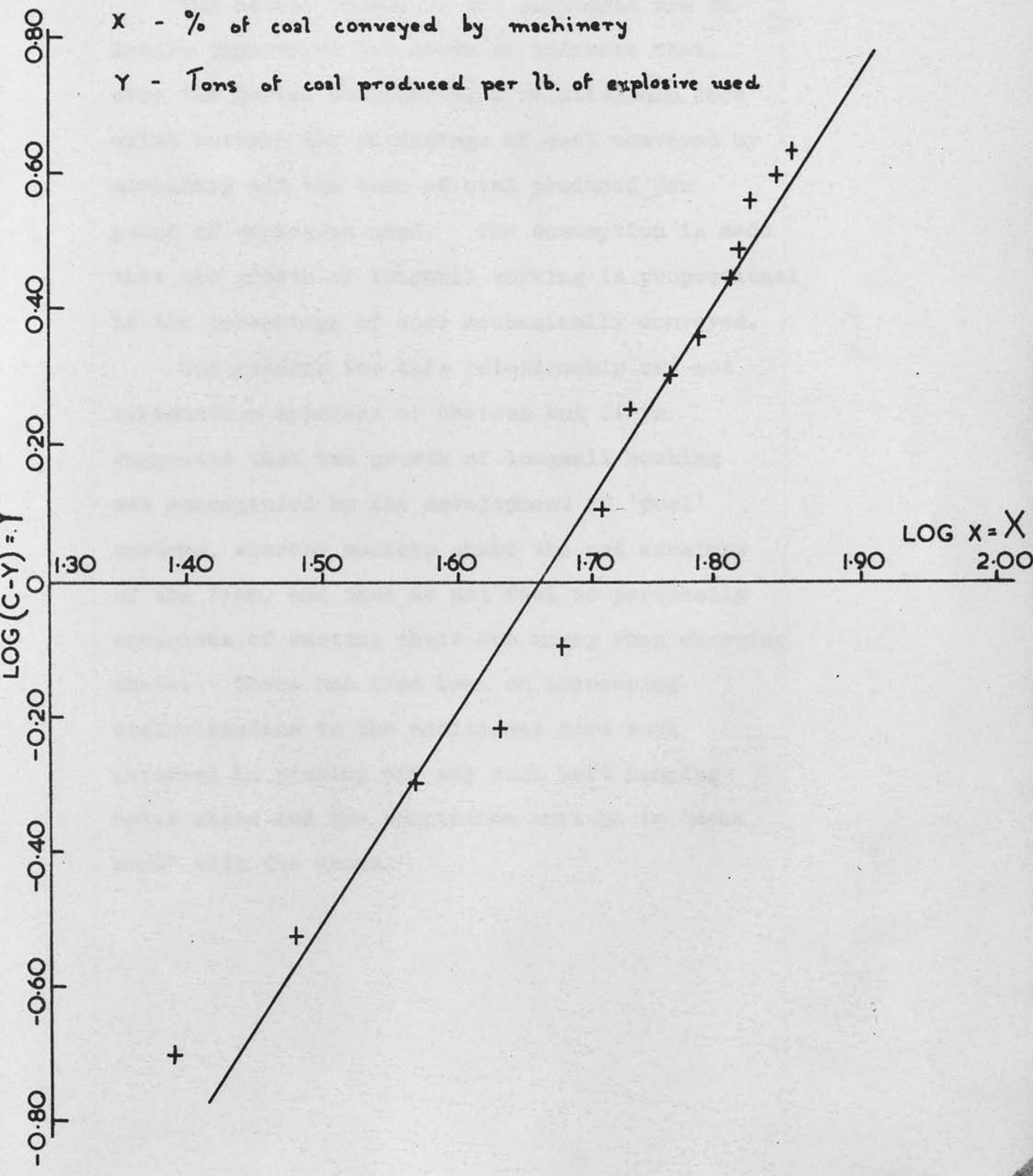
GRAPH IV

THE TONS OF COAL PRODUCED PER POUND OF OF EXPLOSIVE USED, PLOTTED AGAINST THE PERCENTAGE OF COAL CONVEYED BY MACHINERY, FOR THE YEARS 1932 - 1945



GRAPH V

X = LOG X PLOTTED AGAINST
Y = LOG (C-Y)



curve is found to be

$$-y = -9.4 + \left[6.46 \times 10^{-6}\right](x)^{3.13}$$

$$\underline{\text{i.e.}}\quad y = 9.4 - \left[6.46 \times 10^{-6}\right](x)^{3.13}$$

The actual values of the constants are of little importance but serve to indicate that, over the period considered, a relationship does exist between the percentage of coal conveyed by machinery and the tons of coal produced per pound of explosive used. The assumption is made that the growth of longwall working is proportional to the percentage of coal mechanically conveyed.

The reasons for this relationship are not immediately apparent or obvious but it is suggested that the growth of longwall working was accompanied by the development of 'pool' systems, whereby workers share the net earnings of the team, and thus do not feel so personally conscious of wasting their own money when charging shots. There has also been an increasing disinclination to the additional hard work involved in picking off any coal left hanging after shots and the temptation must be to 'make sure' with the shots.

CHAPTER IV

STATISTICAL STUDY OF THE DISTRIBUTION OF SHOTFIRING ACCIDENTS

Any investigator studying shotfiring accidents is aided considerably by the fact that all such accidents, by reason of their nature and irrespective of severity, must be reported to the Mines Inspectorate, and a complete record is kept both by the Ministry of Power and the National Coal Board. Figures relating to mines operated by the National Coal Board are more readily available than those referring to all mines, and are used in this work wherever practicable, as over 90% of explosive is used and resultant accidents occur in the former class. Again, although the more general tables and statistics used and developed apply to the national figures, of necessity closer investi:gation has been largely confined to the Scottish Division and even more particularly to the Lothians Area.

The study of shotfiring accidents, in common with many other types, is complicated by the comparative infrequency of occurrence, indicated by the national average over the period 1934-56 of 3.647 accidents per million shots. For this reason alone it is obviously impossible to compare realistically two collieries on an accident rate basis alone, as even the largest collieries do not fire one million shots per year, and wide variations, which may be meaningless when statistically analysed, may be shown.

It/

It is necessary, however, to have a basis from which a reasonable figure or expectation may be worked out for a group of collieries. Shotfiring accident rates may be stated in several ways.

(1) Accidents per man shifts or man hours. This is the common basis in industry and is, in fact, suitable for comparing the overall risk in different industries but is open to several objections when applied to this branch of mining work.

(a) Not all workers are subject to the risks attendant on shotfiring. Generally, only those men employed on face work, or haulage work near the face, are exposed and not all faceworkers are present on the shift or shifts when shot-:firing takes place.

(b) In recent years great efforts have been made to reduce the proportion of manpower in collieries engaged on unproductive work not at the coal face. In 1948 all underground operations, excluding coal face work, required 344 manshifts per 1,000 tons of output, but by 1955 this figure had been reduced to 326. If the manpower thus freed is employed at the face, then the proportion of men exposed to risk will increase, and logically the figure of accidents per man shifts would also increase. Without closer examination it is impossible to tell whether/

whether any significant change in care in the use and handling of explosives has taken place.

(c) Over the same period (1948-55) intensive efforts were made to introduce face mechanisation wherever possible to make the most efficient use of the skilled face labour available. Many forms of power loading eliminate shotfiring over the complete length of face, with the exception of stable holes, and most reduce it considerably compared with orthodox hand filled faces. Thus, many manshifts will be worked with little or no risk of shotfiring accident and this should logically reduce the figure of accidents per manshifts, tending to cancel out the effect of (b) above. Again the true influence of this factor would be extremely difficult to assess analytically.

(2) Accidents per million tons of output. Supporters of this criterion maintain that, if the figure of accident per million tons is kept to a minimum, then the accident situation is incapable of improvement. This, again, is suitable for comparing the overall accident position of collieries or groups of collieries but is unacceptable in the present analysis for the reasons give in 1 (a), (b) and (c).

(3) Accidents per million shots. This basis is unaffected by the disadvantages of the other two given above and also takes into account the possible safety value of certain techniques of shotfiring/

shotfiring, e.g., simultaneous and delay blasting, where several shots are fired at the same time and detonate together or within a short interval of time. It is thus used for the calculation of expectations in the following analyses.

(a) The Distribution of Shotfiring Accidents over the Period 1934 - 1956

Logically, the first step to be taken in the examination of the distribution of shotfiring accidents is to determine whether there has been any significant change in the national position in the years examined. When the accident rate is plotted on Graph VI, very sharp maximum and minimum values are shown. It is impossible by simple examination of this plot to determine the significance of these high and low values as considerable variations are to be expected on purely statistical grounds. To determine which, if any, variations are significant it is necessary to make certain assumptions. Thus, if the accident situation had in fact remained unchanged, the number of accidents to be expected in any year would be given by:

$$E = N\frac{\Sigma O}{\Sigma n}$$

where, E = expectation for any year

N = number of shots fired in that year

ΣO = total number of accidents in the period considered

Σn/

TABLE III

Accidents, Shots, Accidents/Million Shots and Calculated Expectation for Years 1934 - 56.

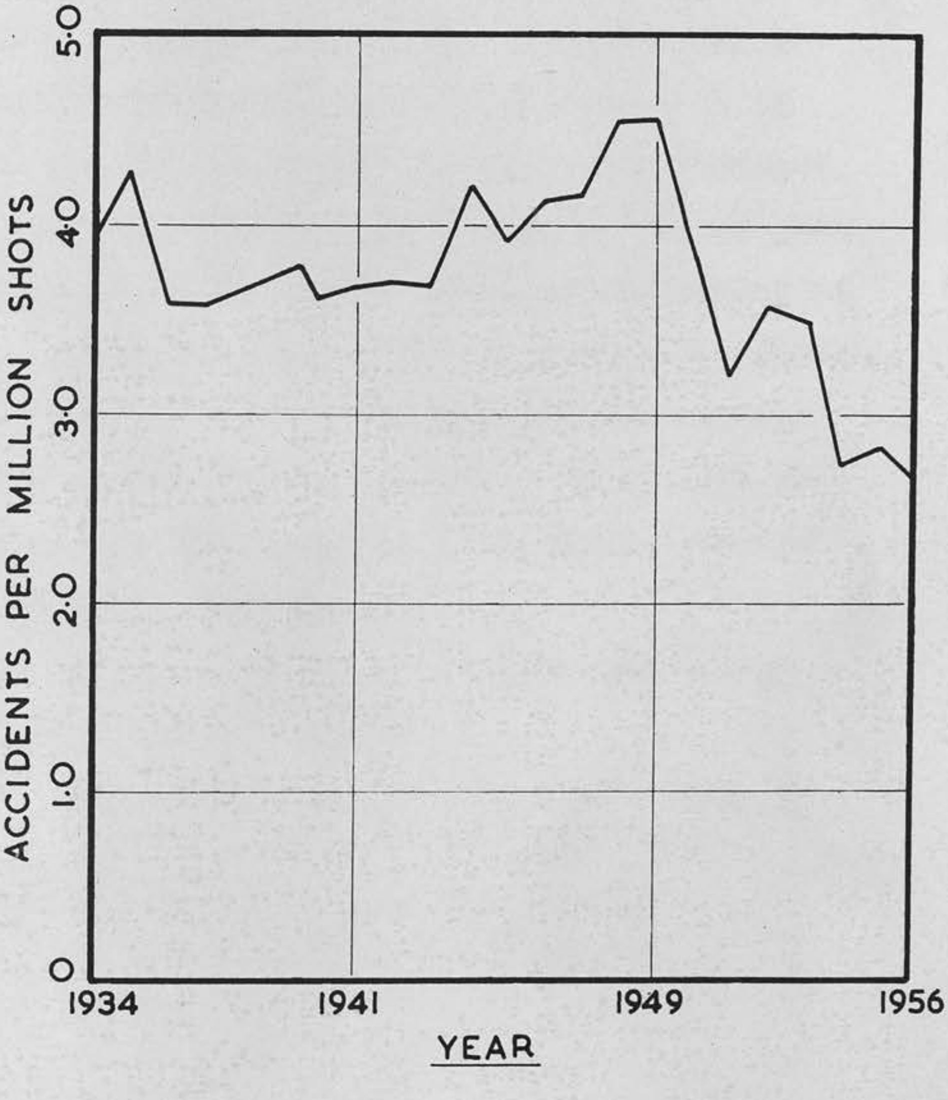
Year	Observed Accidents (O)	Shots fired (millions) (n)	Accidents/ Million Shots	Expectation (E)
1934	206	53.28	3.87	194.0
1935	233	54.37	4.29	198.0
36	210	58.66	3.58	214.0
37	226	63.04	3.59	230.0
38	232	63.71	3.64	232.5
39	237	65.76	3.61	240.0
1940	240	66.64	3.60	243.0
41	243	64.49	3.78	236.5
42	247	66.83	3.69	243.5
43	247	67.34	3.67	245.5
44	285	67.85	4.20	247.0
1945	259	66.47	3.90	242.0
46	295	71.75	4.12	261.5
47	310	74.63	4.16	272.0
48	365	78.62	4.64	286.5
49	386	82.99	4.66	302.0
1950	325	85.71	3.79	312.5
51	292	91.16	3.20	332.0
52	336	94.97	3.54	346.0
53	333	96.92	3.44	353.0
54	271	99.38	2.73	362.0
1955	279	99.41	2.81	362.5
56	268	100.67	2.66	367.0
	$\Sigma O = 6325$	$\Sigma n = 1734.65$		

Notes: (1) Figures from the Annual Reports of H.M. Chief Inspector of Mines

(2) Accident figures refer to total number killed and injured.

GRAPH VI

ACCIDENT RATE PER MILLION SHOTS FOR MINES UNDER THE 1911 COAL MINES ACT [1934-1956]



Σn = total number of shots fired in the same period

$\frac{\Sigma O}{\Sigma n}$ is found to have a value of 3.647 accidents per million shots for the period 1934 - 56.

From this table the observed number of accidents or observation (O) may be compared with the calculated expectation (E). The χ^2 (chi-squared) test may be used to determine the probability that the variation between these numbers is due to chance fluctuations by evaluating $\chi^2 = \frac{(O - E)^2}{E}$ for each year, and comparing the results obtained with statistical tables giving values of χ^2 corresponding to different probabilities. In practice, it is more convenient to plot the parabolae obtained by solving $\frac{(O - E)^2}{E} = \chi^2$ with χ^2 being given values of 2.706, 5.413, 9.550, corresponding to probabilities of 90%, 98% and 99.8% respectively, on a plot of observation against expectation. These curves, called confidence limits, cut the graph into seven regions, into one or other of which each year must fall.

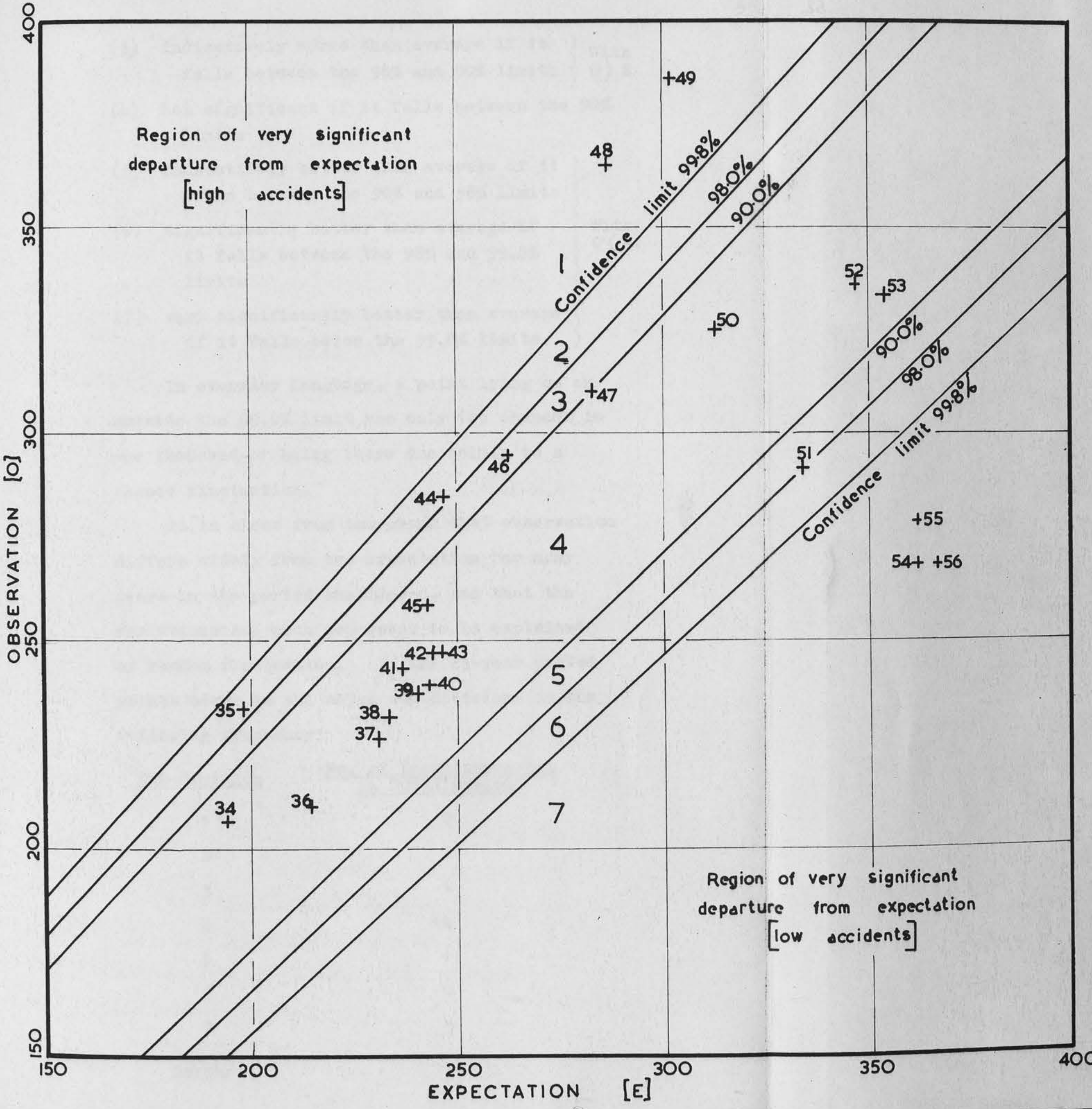
In the customary language of statistics any point is said to be:

(1) very significantly worse than average if it falls above the 99.8% limit

(2) significantly worse than average if it falls between the 99.8% and 98% limits.

$\Bigg\}$ With $O > E$

(3)/



GRAPH VII

ACCIDENTS RESULTING FROM USE OF EXPLOSIVES —— OBSERVATION PLOTTED AGAINST CALCULATED EXPECTATION FOR EACH YEAR OF THE 23-YEAR PERIOD 1934-56

Figures from the annual reports of H.M. Chief Inspector of Mines

(3) indicatively worse than average if it falls between the 98% and 90% limits — With $O > E$

(4) not significant if it falls between the 90% limits

(5) indicatively better than average if it falls between the 90% and 98% limits

(6) significantly better than average if it falls between the 98% and 99.8% limits

(7) very significantly better than average if it falls below the 99.8% limits.

With $O < E$

In everyday language, a point lying on or outside the 99.8% limit has only two chances in one thousand of being there due solely to a chance fluctuation.

It is clear from the graph that observation differs widely from the expectation for many years in the period considered, and that the variations are much too great to be explained by random fluctuation. In the 23-year period points occur in the seven sub-divisions in the following frequency:

<u>Sub-division</u>	<u>No. of Years occurring in Sub-division</u>
1	2
2	2
3	1
4	14
5	1
6	0
7	3

Graph/

Graph VII shows that the accident position has not remained even approximately constant, and that the initial assumption on which the calculated expectations were based was hardly justified. Nevertheless, certain very definite trends may be seen. After a period of 9 years from 1934 - 43 in which time only one year fell outside the 'average' region, a period of deterioration from 1944 - 49 culminated in two exceptionally bad years of 1948 - 49. In January 1948, as a result of the high number of accidents occurring in the use and handling of explosives, a committee was appointed by the Minister of Fuel and Power 'to consider the precautions necessary to secure safety in the use of explosives in coal mines, and in particular to recommend in what way the effective exercise of these precautions can best be ensured in practice'. The report was published in May, 1950, and resulting from its recommendations a new Explosives in Coal Mines Order was enacted in 1951 to simplify and clarify many aspects of the old Order.

It is interesting to trace the trend for the years immediately before and after the very significantly bad years of 1948-49. In 1950, the position had improved to average, and this was followed in 1951 by a year indicatively better than average. The average years of 1952-53 were then followed by the very significantly/

significantly better than average years of 1954-55-56, i.e.

1947 - indicatively worse than average

1948 - very significantly worse than average - Committee appointed

1949 - very significantly worse than average

1950 - average - Committee reports

1951 - indicatively better than average - New E.C.M.O.

1952 - average

1953 - average

1954 - very significantly better than average

1955 - very significantly better than average

1956 - very significantly better than average.

Thus, although it would be tempting to say that the improvement was the direct result of new legislation, the figures do not bear this out entirely. The recent satisfactory improvement may certainly be due to the 1951 Explosives in Coal Mines Order but an improvement was shown from the very bad years of 1948-49 before this came into force. It is suggested that the position had become so bad after years of gradual deterioration that a sudden tightening up became not only advisable but essential, and that this resulted in a voluntary improvement before the new legislation was enforced. In addition, as most shotfiring accidents are caused by projected material, and as the 1951 Explosives in Coal Mines Order contained no new proposals to alleviate/

alleviate this danger, the improvement must have been due, at least in part, to other factors.

(b) The Geographic Distribution of Shotfiring Accidents

The second step to be taken in accident analysis is to examine their geographic distribution and occurrence, and a different statistical technique is required. Thus, when comparing two accident figures x_1, x_2, resulting from the firing of shots n_1, n_2, a two by two contingency table is formed. (ii)

x_1	$n_1 - x_1$	n_1
x_2	$n_2 - x_2$	n_2
$x_1 + x_2$	$n_1 + n_2 - x_1 - x_2$	$n_1 + n_2$

and where every number in the table exceeds 20, the table may be evaluated for χ^2, and from tables the probability that the two figures are representative of the same population may be found.

The value of χ^2 is

$$\frac{\left[x_1(n_2 - x_2) - x_2(n_1 - x_1)\right]^2 (n_1 + n_2)}{n_1 n_2 (x_1 + x_2)(n_1 + n_2 - x_1 - x_2)} \quad \text{(iii)}$$

which may be rewritten

$$\frac{\left[x_1\left(1 - \frac{x_2}{n_2}\right) - x_2 \frac{n_1}{n_2}\left(1 - \frac{x_1}{n_1}\right)\right]^2 \left(1 + \frac{x_1 + x_2}{n_1 + n_2 - x_1 - x_2}\right)}{\frac{n_1}{n_2}(x_1 + x_2)}$$

and as n_1, n_2 are invariably very large in comparison with x_1, x_2 in the study of shotfiring accidents

$$\chi^2 = \frac{\left(x_1 - \frac{n_1}{n_2} x_2\right)^2}{\frac{n_1}{n_2}(x_1 + x_2)}$$

In/

In this work $\frac{n_1}{n_2} = k$ is the ratio of the number of shots fired in the areas under examination, and is regarded as the rates of anticipated probability of accidents in the two samples.

$$\chi^2 = \frac{(x_1 - kx_2)^2}{k(x_1 + x_2)}$$

In the analysis that follows, each National Coal Board Division is compared in turn with the rest of the country and the symbols used are as follows:

x_1	=	Accidents in Division
$x_1 + x_2$	=	National total of accidents
N	=	Number of shots fired in Division
T	=	Number of shots fired in country
k	=	Ratio of number of shots fired in Division to number of shots fired in rest of country, i.e. $\frac{N}{T - N}$

Confidence limits are obtained by plotting $k(x_1 + x_2)$ against $(x_1 - kx_2)$ for the values of χ^2 given i.e. 2.706, 5.413 and 9.550. To enable trends in different divisions to be traced, one graph is prepared for each Division and a point plotted for each year.

Tables IV - XI. The numbers of shots fired in N.C.B. mines by Divisions, the observed numbers of accidents and the calculated expectation for the years 1950-57 inclusive.

TABLE IV/

TABLE IV - YEAR 1950

Division	N	T - N	x_1	x_2	kx_2	$x_1 - kx_2$	$k(x_1 + x_2)$
Scottish	13.220	69.940	67	239	45.3	21.7	58.0
Northern	8.166	74.994	25	281	30.6	- 5.6	33.4
Durham	11.714	71.446	54	252	41.2	12.8	50.2
North Eastern	14.761	68.399	37	269	58.0	-21.0	66.0
North Western	8.448	74.712	25	281	31.9	- 6.9	34.7
East Midlands	15.384	67.776	53	25.3	57.5	- 4.5	69.4
West Midlands and South East	7.865	75.295	30	276	28.7	1.3	31.9
South Western	3.602	79.558	15	291	13.2	1.8	13.9

T - 83.160 $x_1 + x_2$ - 306

TABLE V - YEAR 1951

Division	N	T - N	x_1	x_2	kx_2	$x_1 - kx_2$	$k(x_1 + x_2)$
Scottish	13.555	73.949	54	220	40.3	13.7	50.2
Northern	8.328	79.176	20	254	26.7	- 6.7	28.7
Durham	12.157	75.347	45	229	36.8	8.2	44.0
North Eastern	15.693	71.811	42	232	50.9	- 8.9	60.0
North Western	8.999	78.505	23	251	28.7	- 5.7	31.4
East Midlands	16.973	70.531	47	227	54.3	- 7.3	65.7
West Midlands and South East	8.021	79.483	32	242	24.4	7.8	27.6
South Western	3.778	83.726	11	263	11.9	- 0.9	12.3

T - 87.504 $x_1 + x_2$ - 274

TABLE VI - YEAR 1952

Division	N	T - N	x_1	x_2	kx_2	$x_1 - kx_2$	$k(x_1 + x_2)$
Scottish	13.896	76.879	67	257	46.5	20.5	58.5
Northern	8.536	82.239	31	293	30.5	0.5	33.6
Durham	12.386	78.389	50	274	43.2	6.8	51.0
North Eastern	16.740	74.035	40	284	64.1	-24.1	73.1
North Western	9.491	81.284	23	301	35.1	-13.1	37.8
East Midlands	17.629	73.146	61	263	63.5	- 2.5	78.0
West Midlands and South East	8.692	82.083	39	285	30.1	8.9	34.2
South Western	3.828	86.947	13	311	12.3	0.7	12.8

T - 90.775 $x_1 + x_2$ - 324

TABLE VII - YEAR 1953

Division	N	T - N	x_1	x_2	kx_2	$x_1 - kx_2$	$k(x_1 + x_2)$
Scottish	14.297	78.705	86	236	43.0	43.0	58.5
Northern	8.680	84.322	25	297	30.5	- 5.5	33.1
Durham	12.223	80.779	49	273	41.4	7.6	48.9
North Eastern	17.276	75.726	31	291	66.5	-35.5	73.5
North Western	9.833	83.189	31	291	34.4	- 3.4	38.0
East Midlands	17.840	75.162	48	274	65.0	-17.0	76.5
West Midlands and South East	9.060	83.942	27	295	31.8	- 4.8	34.7
South Western	3.793	89.209	25	297	12.6	12.4	13.7

T - 93.002 $\quad x_1 + x_2$ - 322

TABLE VIII - YEAR 1954

Division	N	T - N	x_1	x_2	kx_2	$x_1 - kx_2$	$k(x_1 + x_2)$
Scottish	14.671	80.682	67	198	36.0	31.0	48.2
Northern	8.732	86.621	29	236	23.8	5.2	26.7
Durham	12.502	82.851	27	238	36.0	- 9.0	40.0
North Eastern	17.987	77.366	34	231	53.7	-19.7	61.5
North Western	9.606	85.747	21	244	27.4	1 6.3	29.7
East Midlands	18.608	76.745	55	210	51.0	4.0	64.3
West Midlands and South East	9.457	85.896	27	238	26.2	0.8	29.2
South Western	3.789	91.564	5	260	10.7	- 5.7	10.9

T - 95.353 $x_1 + x_2$ - 265

TABLE IX - YEAR 1955

Division	N	T - N	x_1	x_2	kx_2	$x_1 - kx_2$	$k(x_1 + x_2)$
Scottish	14.637	81.031	58	214	38.8	19.2	49.2
Northern	8.826	86.842	20	252	25.7	- 5.7	27.8
Durham	12.886	82.782	30	242	37.7	- 7.7	42.5
North Eastern	17.723	77.945	28	244	55.6	-27.6	62.0
North Western	9.808	65.860	25	247	36.8	-11.8	40.5
East Midlands	18.734	76.934	58	214	52.3	5.7	66.5
West Midlands and South East	9.324	86.344	37	235	25.4	11.6	29.4
South Western	3.730	91.938	16	256	10.3	5.7	10.9

T - 95.668 $x_1 + x_2$ - 272

TABLE X - YEAR 1956

Division	N	T - N	x_1	x_2	kx_2	$x_1 - kx_2$	$k(x_1 + x_2)$
Scottish	14.631	82.292	46	209	37.2	8.8	45.4
Northern	9.061	87.862	22	233	24.0	- 2.0	26.2
Durham	13.154	83.769	29	226	35.5	- 6.5	40.0
North Eastern	18.354	78.569	45	210	48.9	- 3.9	59.5
North Western	9.799	87.124	23	232	26.0	- 3.0	28.6
East Midlands	18.507	78.416	62	193	45.5	16.5	60.2
West Midlands and South East	9.276	87.647	21	234	24.8	- 3.8	27.0
South Western	4.142	91.781	7	248	11.2	- 4.2	11.5

T - 96.923 $\quad x_1 + x_2$ - 255

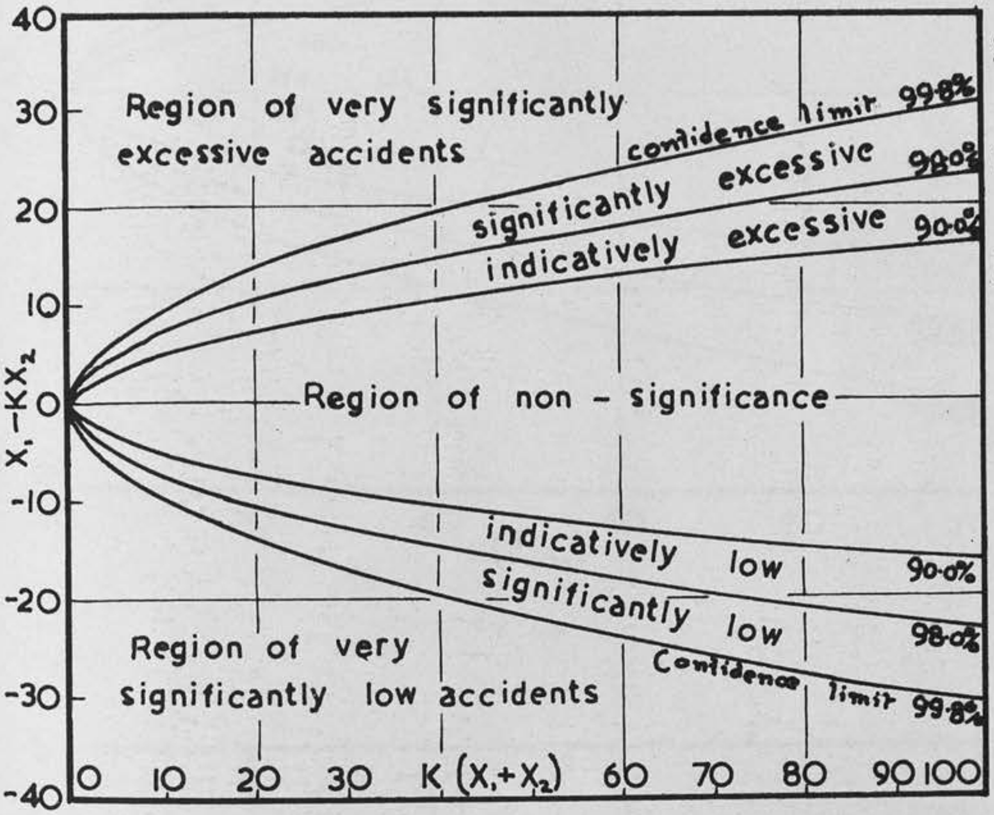
TABLE XI - YEAR 1957

Division	N	T - N	x_1	x_2	kx_2	$x_1 - kx_2$	$k(x_1 + x_2)$
Scottish	14.254	81.334	48	201	35.2	12.8	43.6
Northern	8.915	86.673	24	225	23.2	0.8	25.6
Durham	13.201	82.387	36	213	34.3	1.7	40.0
North Eastern	18.132	77.456	36	213	49.9	-13.9	58.3
North Western	9.964	85.624	15	234	27.1	-12.1	28.9
East Midlands	17.210	78.387	46	203	44.7	1.3	54.8
West Midlands and South East	9.264	86.324	30	219	25.6	4.4	26.6
South Western	4.648	90.940	14	235	12.2	1.8	13.0

T - 95.588 $\qquad x_1 + x_2$ - 249

GRAPHS VIII–XVI SHOTFIRING ACCIDENTS IN NCB MINES 1950–1957

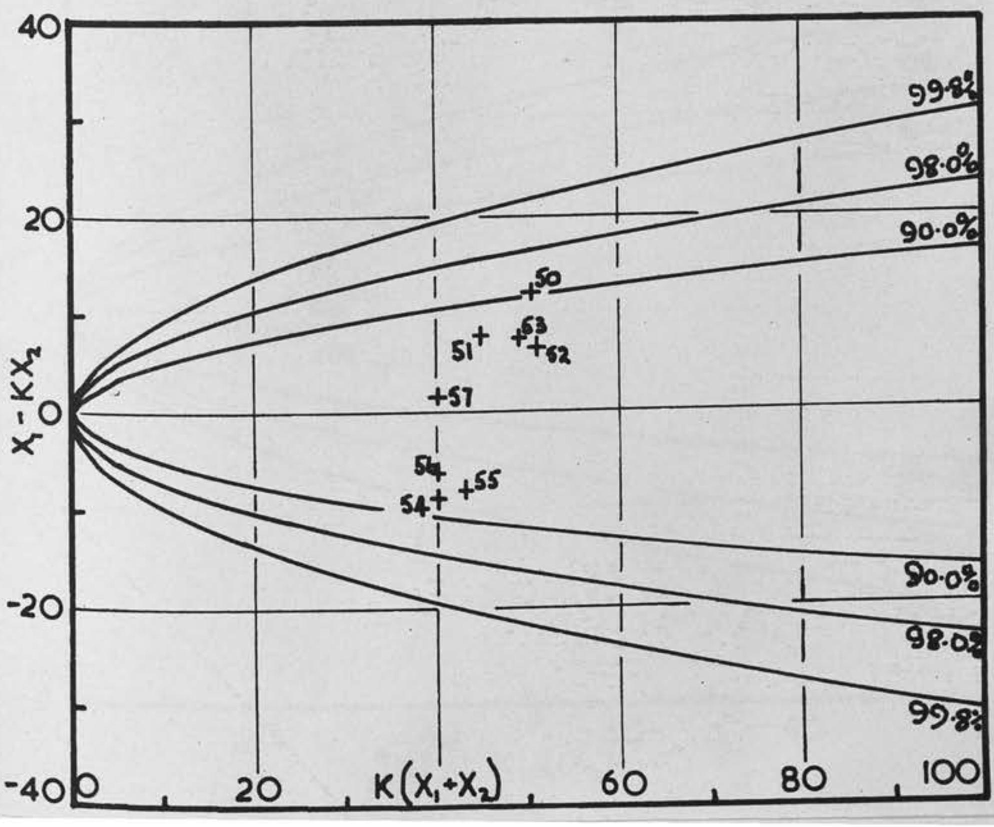
Each Division is compared with the rest of the rest of the country, using the ratio of the numbers of shots fired as the basis of comparison. One point is plotted for each year on the Divisional graph to enable any local trends to be more easily identified. To avoid confusion, only the first graph gives in full the meaning of the symbols used and the significance of the regions on the graphs.



X_1 = NUMBER OF ACCIDENTS IN DIVISION

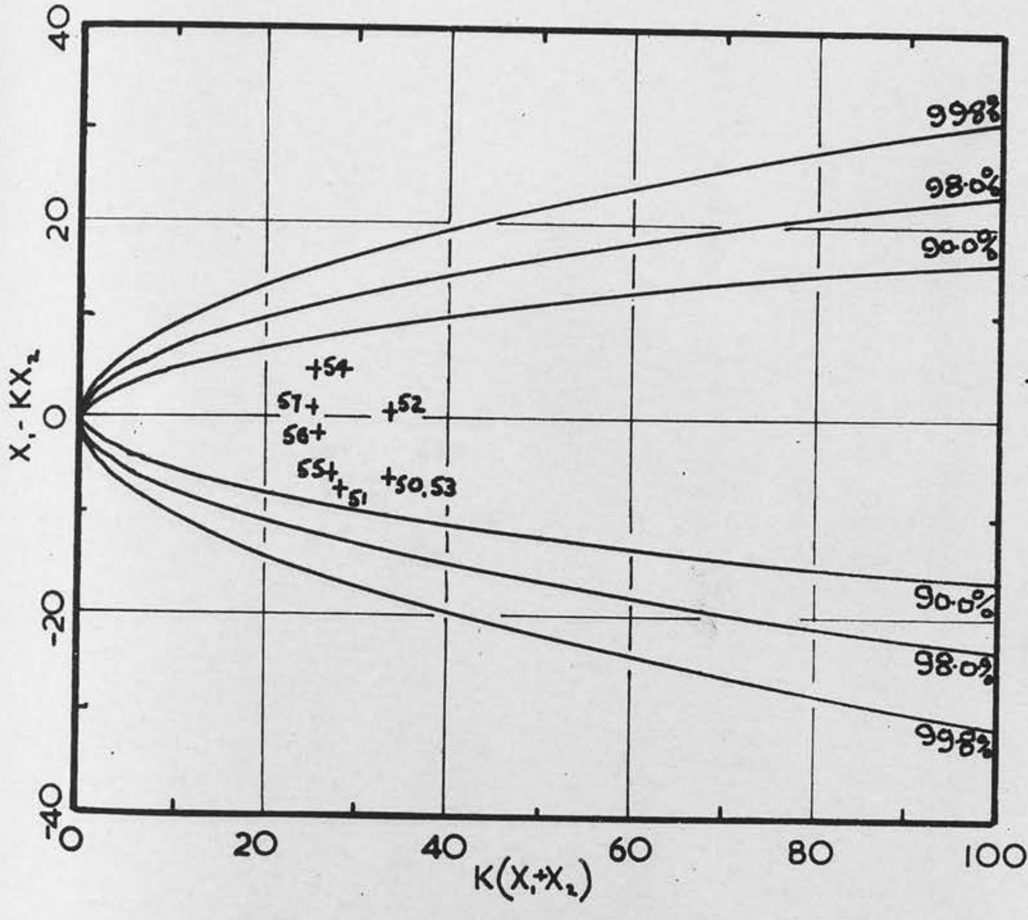
X_2 = NUMBER OF ACCIDENTS IN REST OF COUNTRY

K = RATIO OF NO. OF SHOTS IN DIVISION TO NO. OF SHOTS IN REST OF COUNTRY



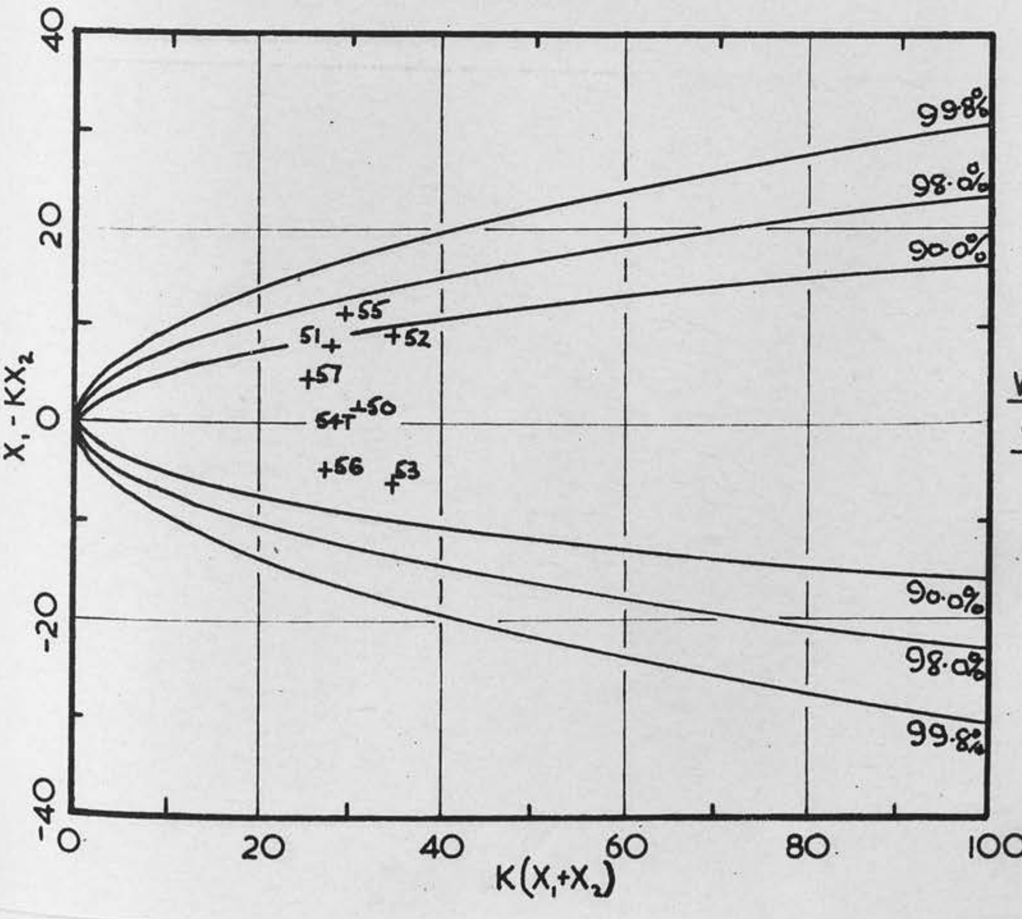
GRAPH VIII

DURHAM DIVISION



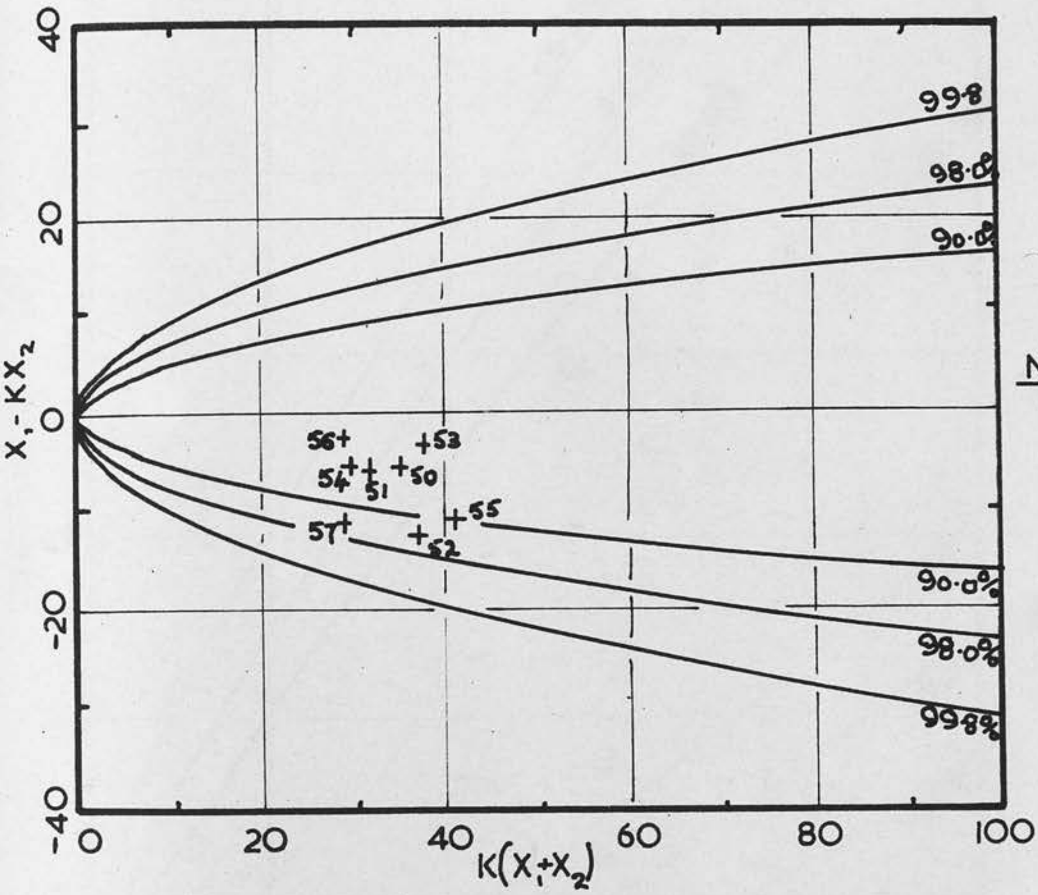
GRAPH IX

NORTHERN DIVISION



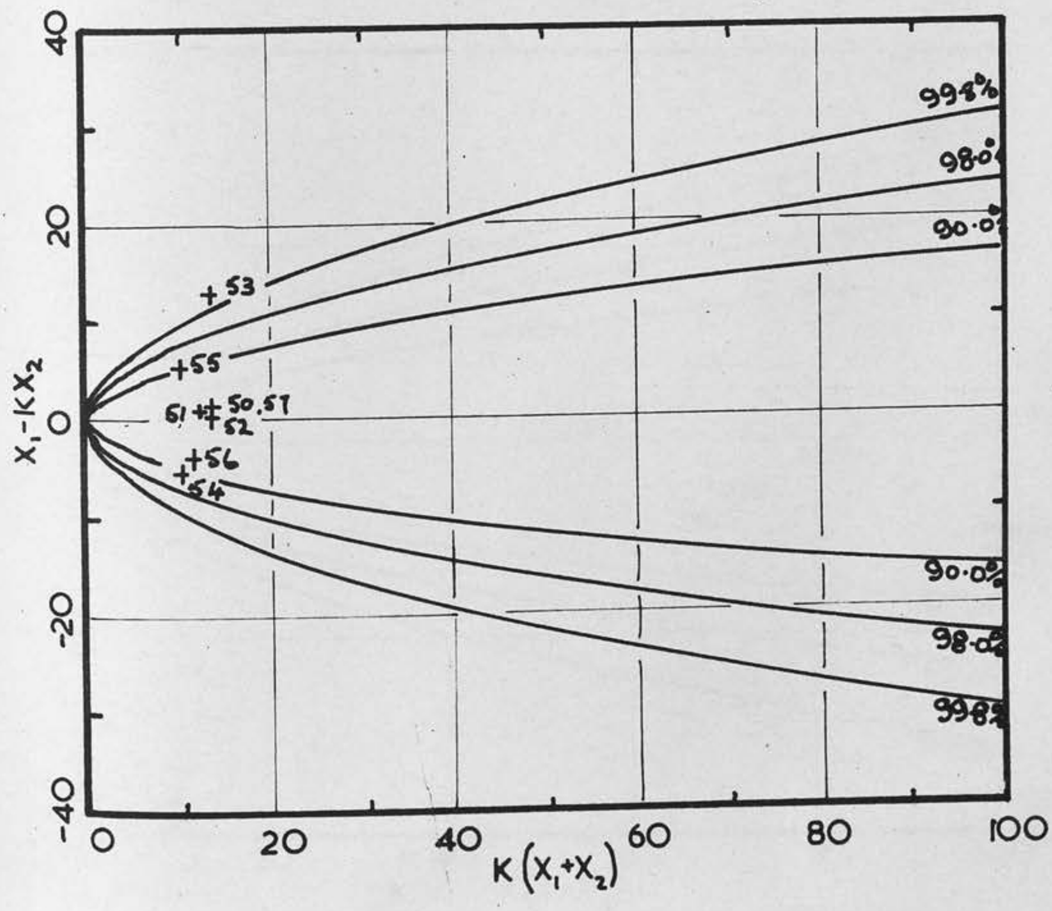
GRAPH X

WEST MIDLANDS AND SOUTH-EAST DIVISIONS



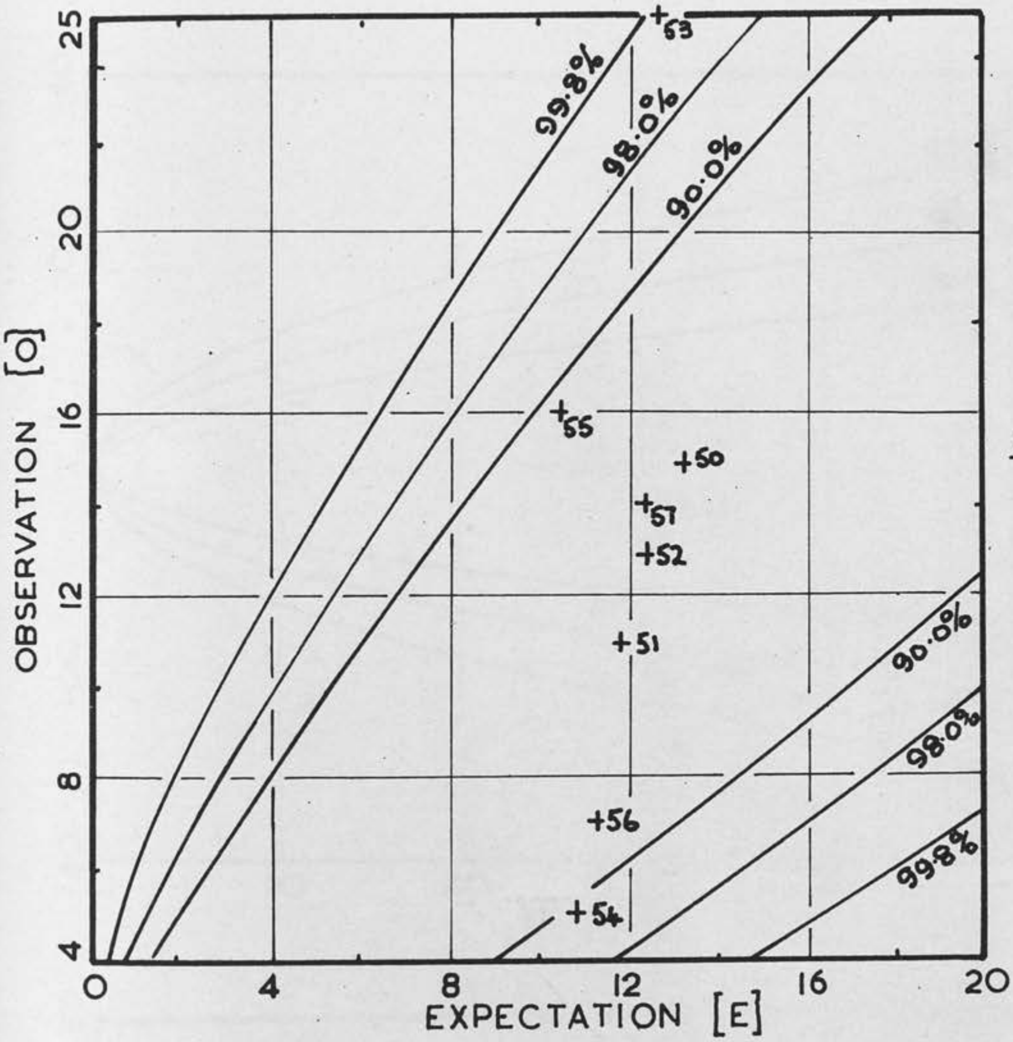
GRAPH XI

NORTH - WESTERN DIVISION



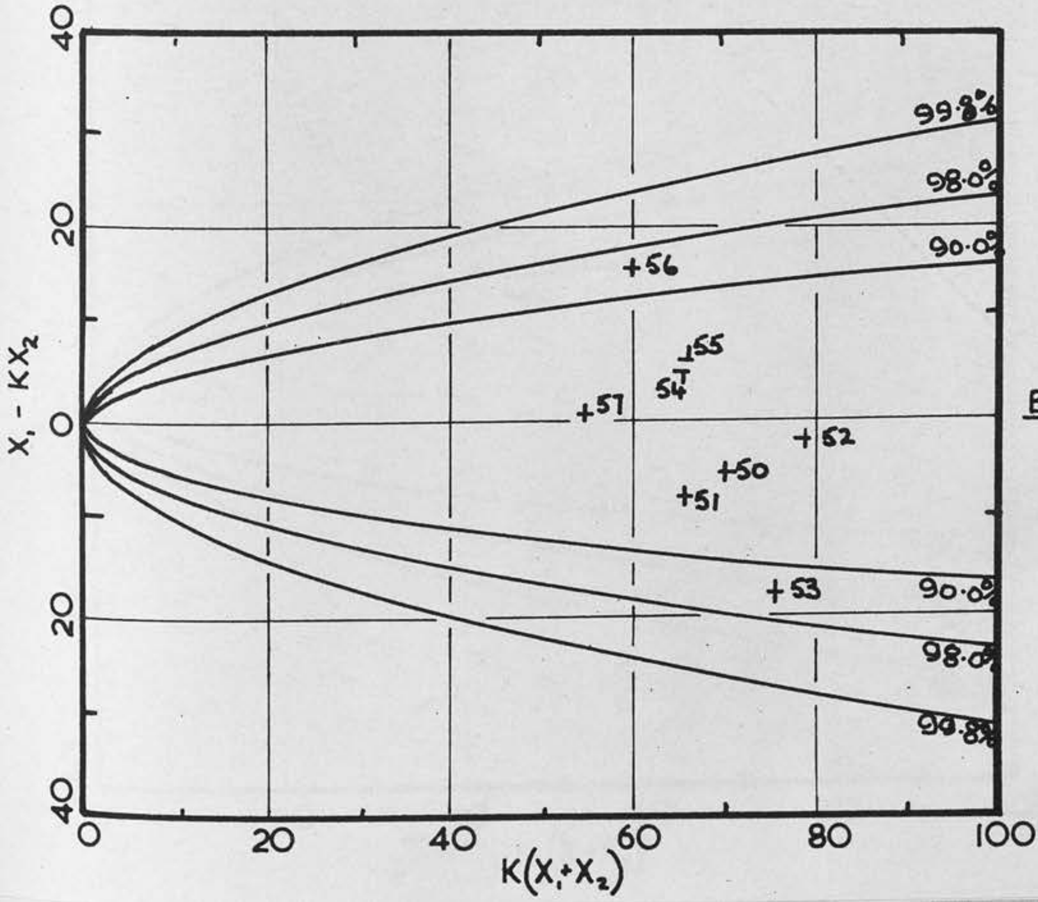
GRAPH XII

SOUTH - WESTERN DIVISION



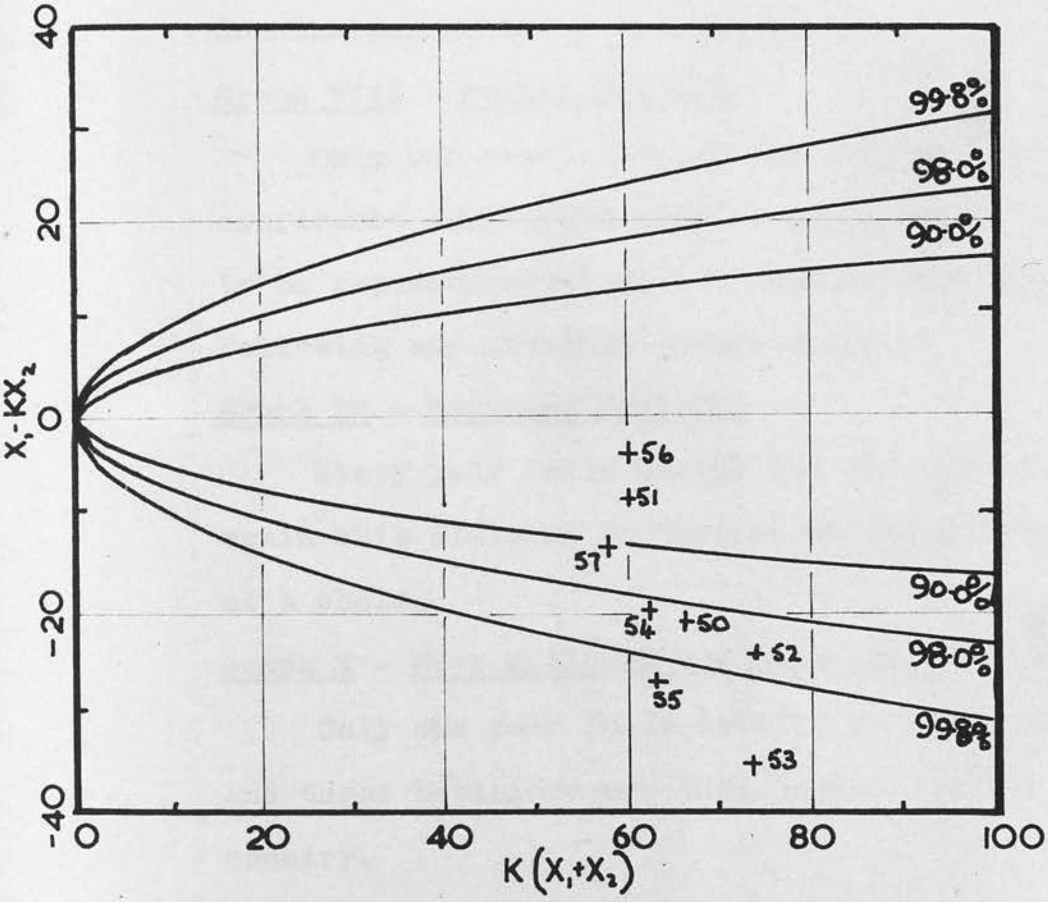
GRAPH XIII

SOUTH-WESTERN DIVISION



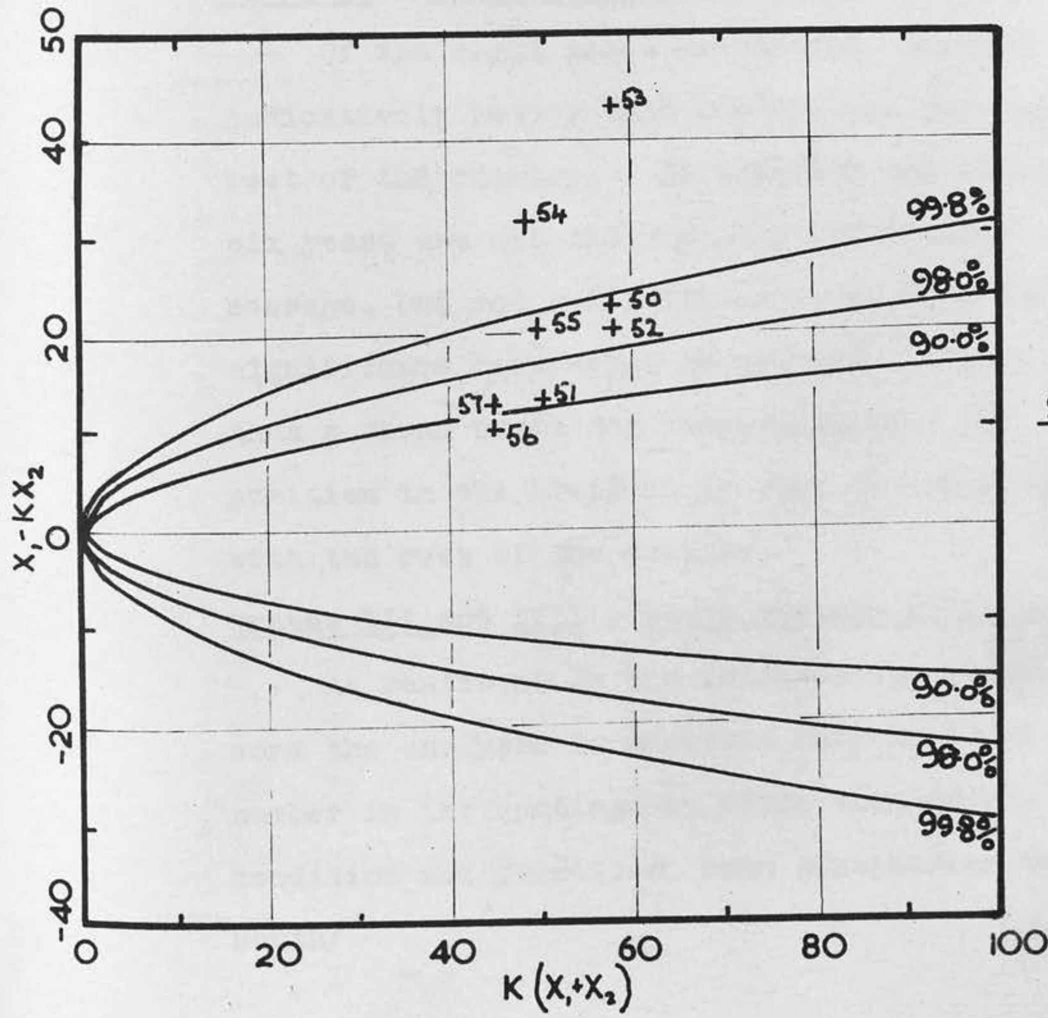
GRAPH XIV

EAST MIDLANDS DIVISION



GRAPH XV

NORTH EASTERN DIVISION



GRAPH XVI

SCOTTISH DIVISION

Conclusions obtained from the Graphs.

Graph VIII - Durham Division

Only one year, 1950, falls outside the 90% confidence limits and this Division may be said to be representative of the country as a whole, following any national trends closely.

Graph IX - Northern Division

Every year falls inside the 90% limits and again this Division is typical of the country as a whole.

Graph X - West Midlands and South East Divisions

Only one year falls outside the 90% limits, and these Divisions are thus typical of the country.

Graph XI - North Western Division

Of the eight years considered, two are indicatively better than the average for the rest of the country. In addition the other six years are all individually better than average, but not sufficiently so to have any significance considered separately. There is thus a sound basis for concluding that the position in the Division is good in comparison with the rest of the country.

Graphs XII and XIII - South Western Division

As mentioned in the Introduction to this work the analysis is accurate only if every number in the contingency table exceeds 20, a condition not fulfilled when considering the South/

South Western Division. The effect of this is to exaggerate the significance of the results obtained, as seen in the plot for the Division, and it is necessary to employ the technique later used for comparison of areas to obtain a true assessment of the accident situation. The revised plot is seen in Graph XIII. It is interesting to note that a significantly bad year in 1953 was followed in 1954 by an indicatively good year in 1954, probably as the result of the compaign which would follow 1953. A period of average years 1955-56-57 followed indicating a return to normal conditions of care and attention.

Graph XIV - East Midland Division

In this division the points are scattered with no suggestion of markedly better or worse conditions than the rest of the country. Two years fall outside the 'non-significant' region of the graph, with one, 1953, being indicatively better than average and the other, 1956, indicatively worse than average. There is a suggestion of a trend towards improvement between 1953 and 1956, but 1957 showed a return to average conditions.

Graph XV - North Eastern Division

In this Division, only two years are classified as average and two years are very significantly/

significantly better than the rest of the country. There is a trend to be followed, as with the East Midlands Division, but in this case a period of improvement in the years 1950-53, excepting 1951, is followed by a deterioration from the very good year of 1953 to the average year of 1956, but an improvement does occur in 1957 to indicatively better than average.

Graph XVI - Scottish Division

In the Scottish Division, only one year, 1956, falls into the average classification and two years, 1953 and 1954, are very significantly worse than average. Scotland has in fact by far the worst record of shotfiring accidents of any Division and although it might tentatively be suggested that some improvement is being shown on the basis of the drop shown from 1953-54, 1954-55, 1955-56, followed by a slight deterioration in 1957, more years will be required before it can be stated that the position has indeed improved.

As Scotland has the worst shotfiring accident record, the logical step is now to continue the sub-division and compare areas to determine which, if any, account for an excessive proportion of the total. As already stated, the χ^2 analysis is inaccurate for small numbers but an observation can be compared with an expectation calculated on the ratio of the number of shots fired in the areas under consideration. Probabilities, and hence/

hence confidence limits, are obtained by making use of the Poisson series which states that the probability P, of observing exactly x accidents when the expectation is E is

$$\frac{e^{-E}E^{x}}{x!} \qquad \text{(iv)}$$

and summing the cumulative terms in the Poisson series we find

$$P(x,E) = \sum_{r=x}^{\infty} \frac{e^{-E}E^{r}}{r!}$$

where P(x,E) is the probability of an observation being equal to or greater than x. Central confidence limits are obtained by assuming that there is an equal chance of observations being significantly high or low.

The 90% confidence levels are obtained by solving $P(x,E) = .05$ and $P(x-1,E) = .95$ for E, for successive integral values of x. Similarly, the 98% levels are obtained by solving $P(x,E) = .01$ and $P(x-1,E) = .99$, and the 99.8% are found by solving $P(x,E) = .001$ and $P(x-1,E) = .999$.

In this examination, expectations for areas are calculated from the shotfiring accident rate of the rest of the Division, i.e.

$$E = N\frac{d - O}{D - N}$$

where, E = the expectation

D = number of shots fired in Division

N = number of shots fired in Area

d = number of accidents in Division

O = number of accidents in Area.

It/

It is unfortunate from the point of view of this investigation that a reorganisation within the Scottish Division resulted in changes in boundaries of the Areas and that a general comparison is not possible for years before 1954.

Tables XII - XV. The numbers of shots fired in Scottish Division by Areas, the observed number of accidents, and the calculated expectation for years 1954 - 1957.

TABLE XII - YEAR 1954

Area	N	D - N	O	d - O	E
West Fife	2.375	12.297	11	56	10.8
Lothians	2.078	12.594	15	52	8.6
Central West	1.856	12.816	3	64	9.3
Central East	2.224	12.448	8	59	10.6
West Ayr	1.280	13.392	4	63	6.05
Alloa	1.138	13.534	10	57	4.8
East Fife	1.601	13.071	9	58	7.1
East Ayr	2.120	12.552	7	60	10.2
	D - 14.672		d - 67		

Table/

TABLE XIII - YEAR 1955

Area	N	D - N	O	d - O	E
West Fife	2.248	12.389	11	45	8.15
Lothians	2.166	12.471	9	47	8.15
Central West	1.707	12.930	8	48	6.35
Central East	2.281	12.356	4	52	9.6
West Ayr	1.340	13.297	5	51	5.15
Alloa	1.165	13.472	3	53	4.6
East Fife	1.707	12.930	4	52	6.85
East Ayr	2.023	12.614	12	44	7.0
	D - 14.637		d - 56		

TABLE XIV - YEAR 1956

West Fife	2.273	12.358	8	38	7.0
Lothians	2.225	12.406	4	42	7.55
Central West	1.655	12.976	7	39	4.95
Central East	2.257	12.374	2	44	8.05
West Ayr	1.271	13.360	5	41	3.90
Alloa	1.108	13.513	3	43	3.55
East Fife	1.609	13.022	8	38	4.7
East Ayr	2.233	12.398	9	37	6.7
	D - 14.631		d - 46		

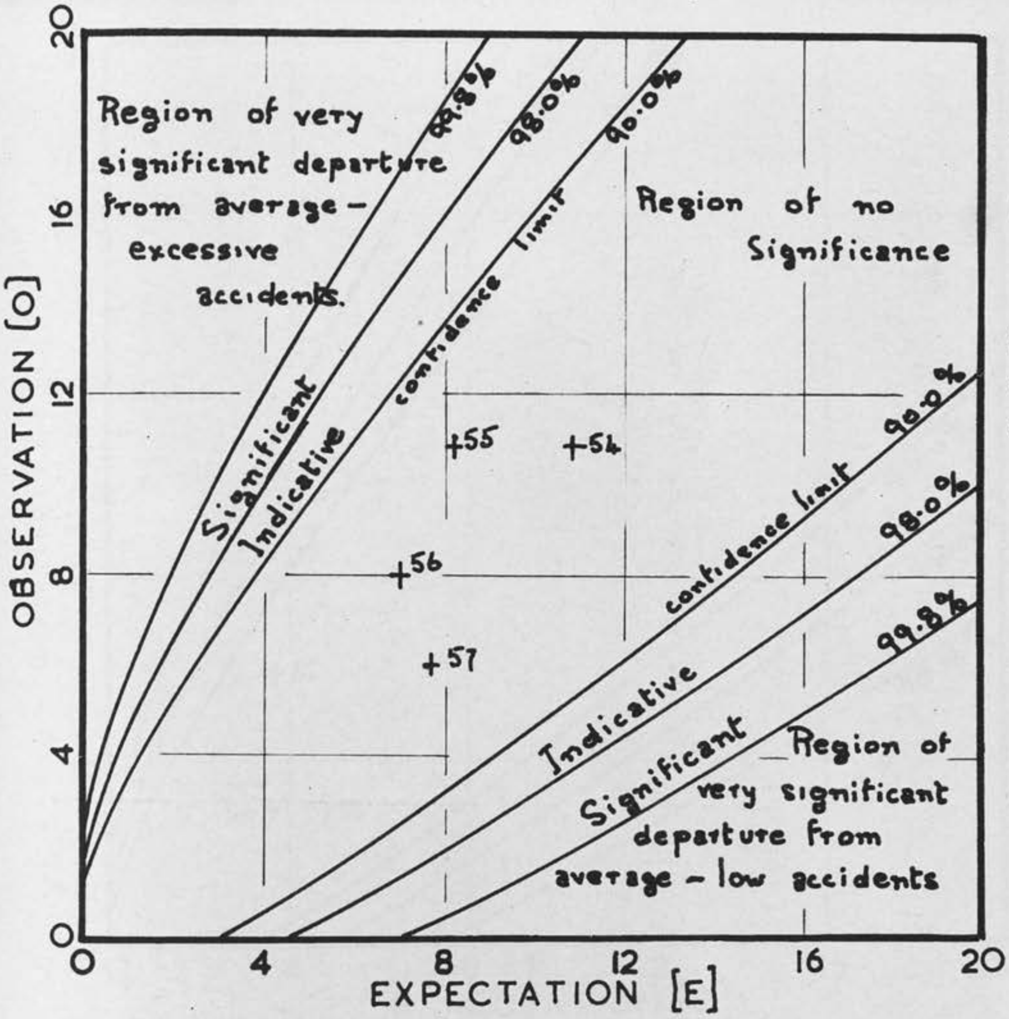
TABLE/

TABLE XV - YEAR 1957

Area	N	D - N	O	d - O	E
West Fife	2.202	12.053	6	42	7.7
Lothians	2.214	12.041	5	43	8.0
Central West	1.595	12.660	6	42	5.3
Central East	2.178	12.077	8	40	7.25
West Ayr	1.227	13.028	2	46	4.3
Alloa	1.152	13.103	6	42	3.7
East Fife	1.558	12.697	7	41	5.05
East Ayr	2.129	12.126	8	40	7.0
	D - 14.255		d - 48		

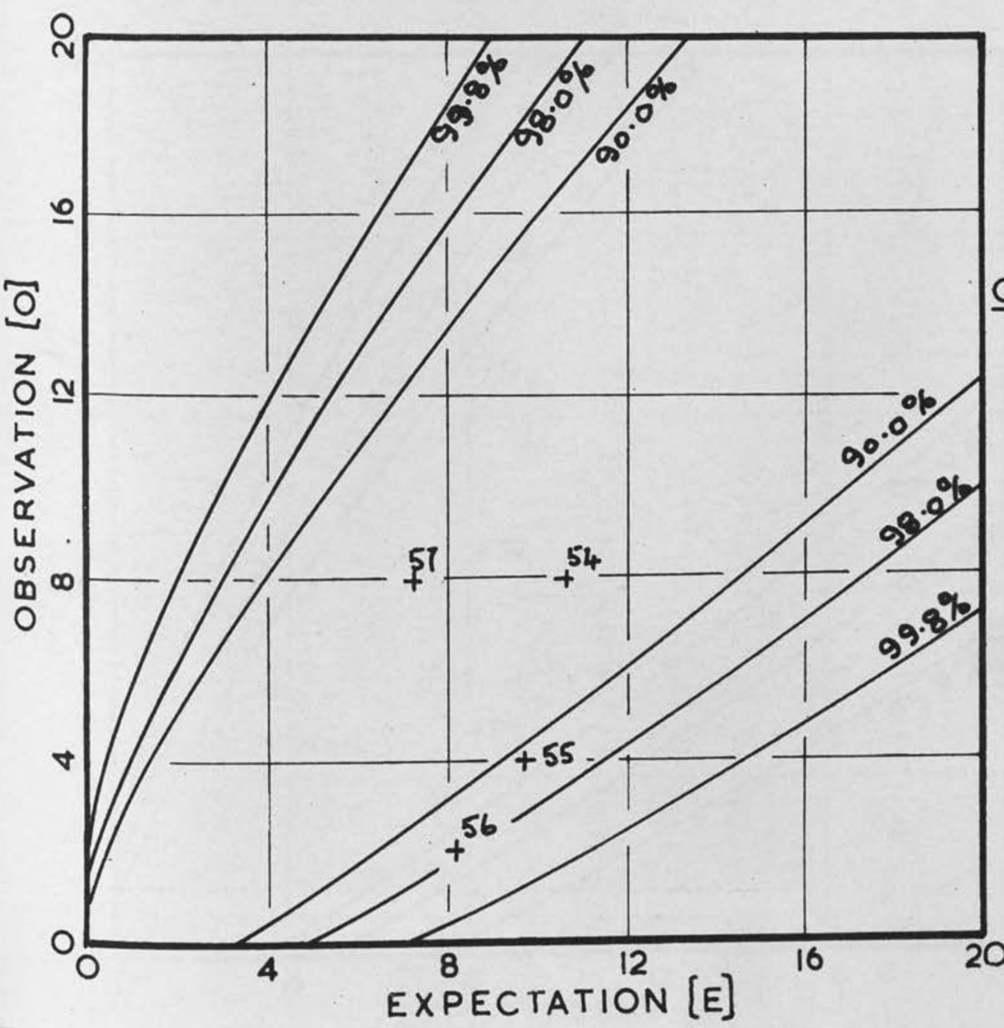
Comments on the plots obtained. In the plots for West Fife, East Fife, East Ayr and West Ayr, on Graphs XVII, XXI, XXIII, XXIV, no variation between observation and expectation unexplainable by random fluctuation occurs, and it may be concluded that these areas follow the national trend closely. The Lothians, Alloa and Central West Areas, Graphs XX, XIX, XXII, each have one year in the indicative regions but only in the Lothians Area is there any suggestion of a trend, in this case, towards improvement. In the remaining Area, Central East, on Graph XVIII, two out of the four years in the period examined are indicatively lower than the average for the other Areas, which serves/

Graphs XVII - XXIV
Shotfiring accidents in N.C.B. mines, Scottish Division 1954-1957. Each area is compared with the rest of the division, using the ratio of the numbers of shots fired as the basis of comparison.



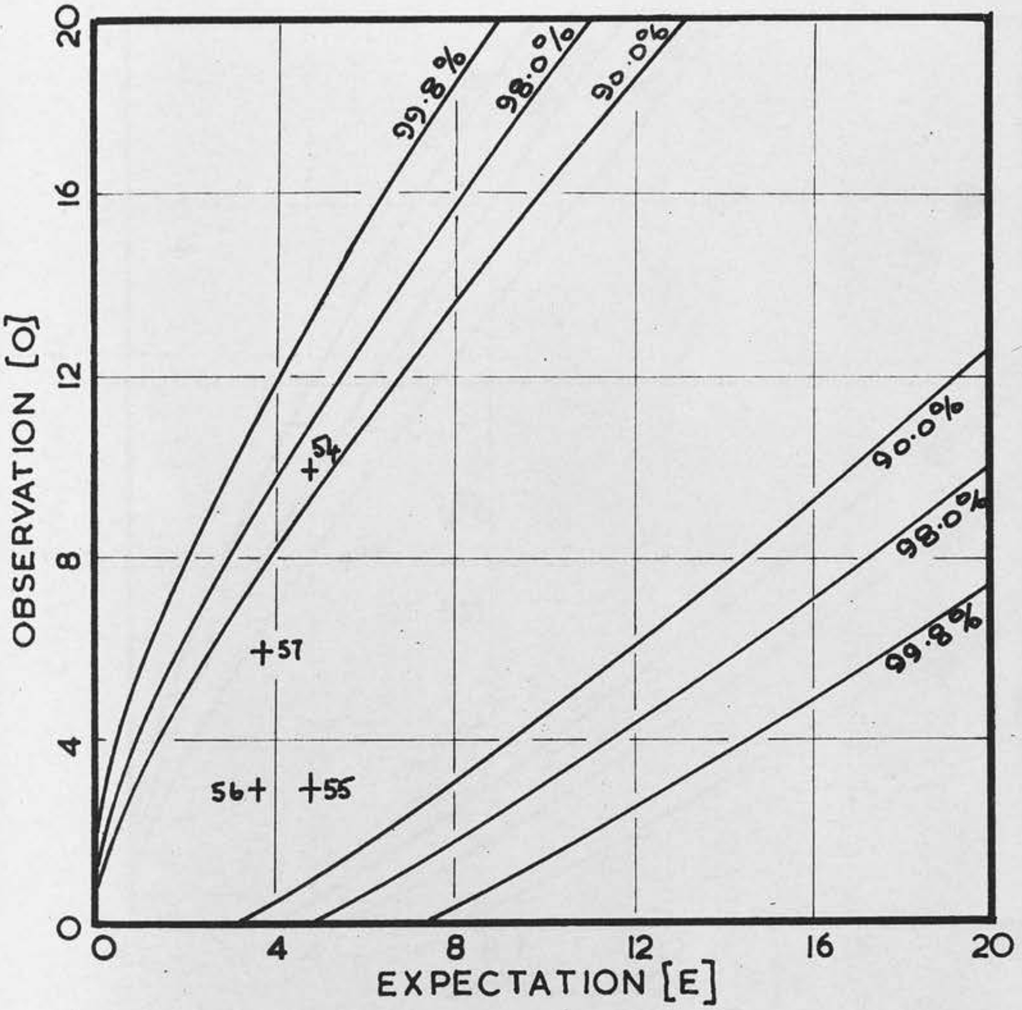
GRAPH XVII

WEST FIFE AREA



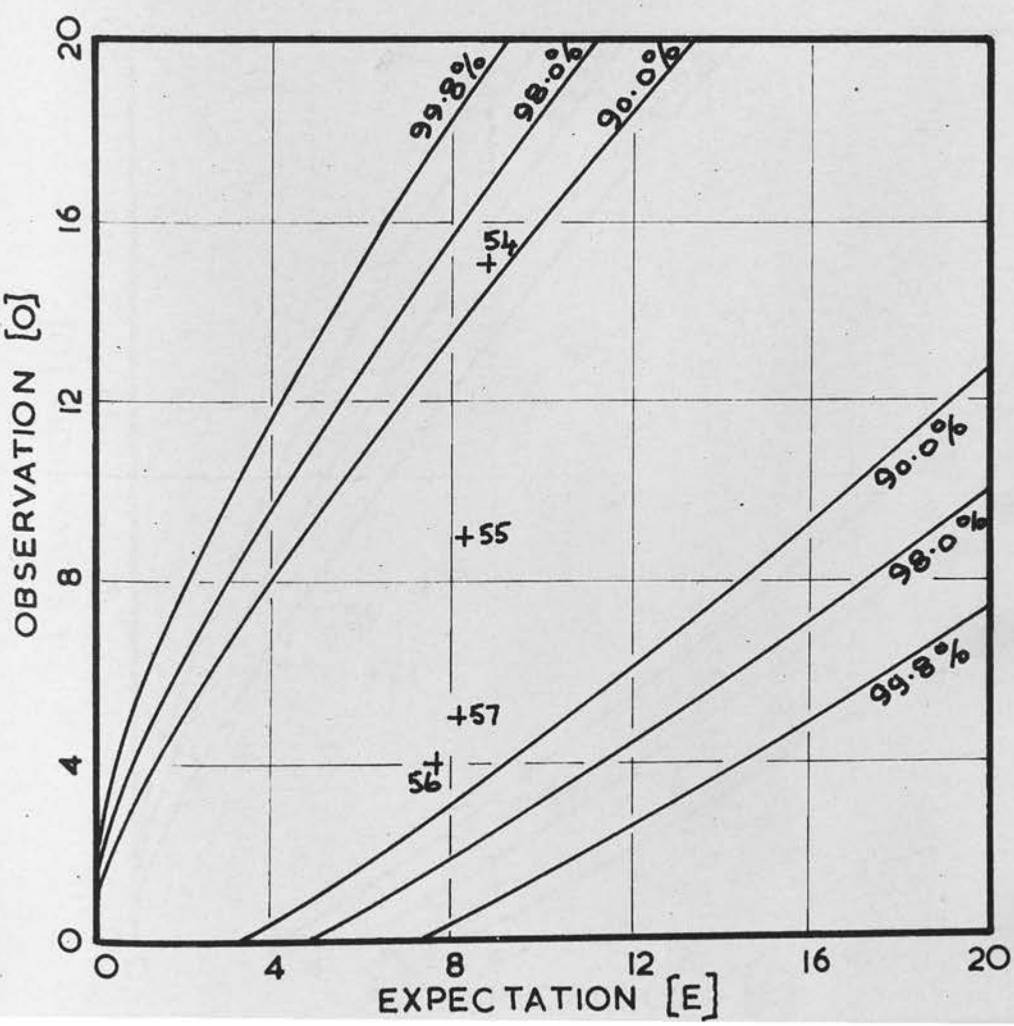
GRAPH XVIII

CENTRAL EAST AREA



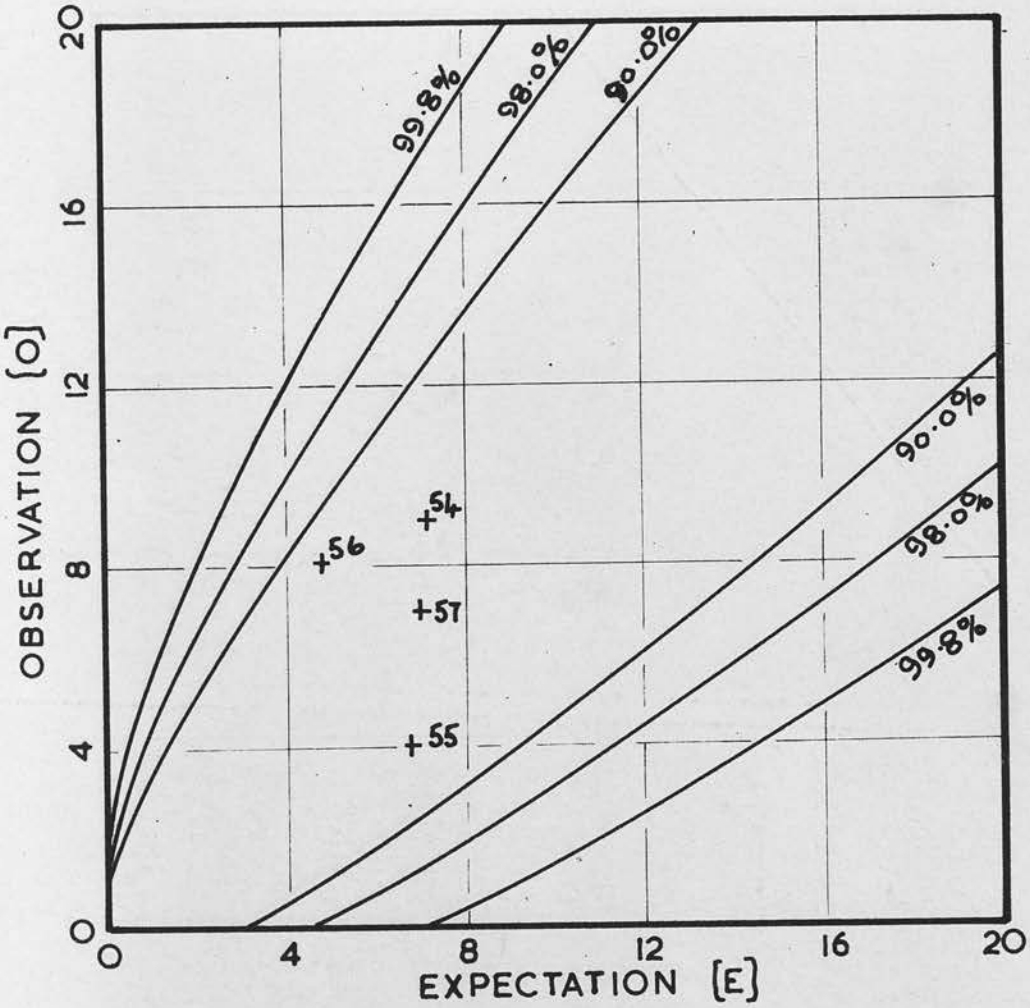
GRAPH XIX

ALLOA AREA



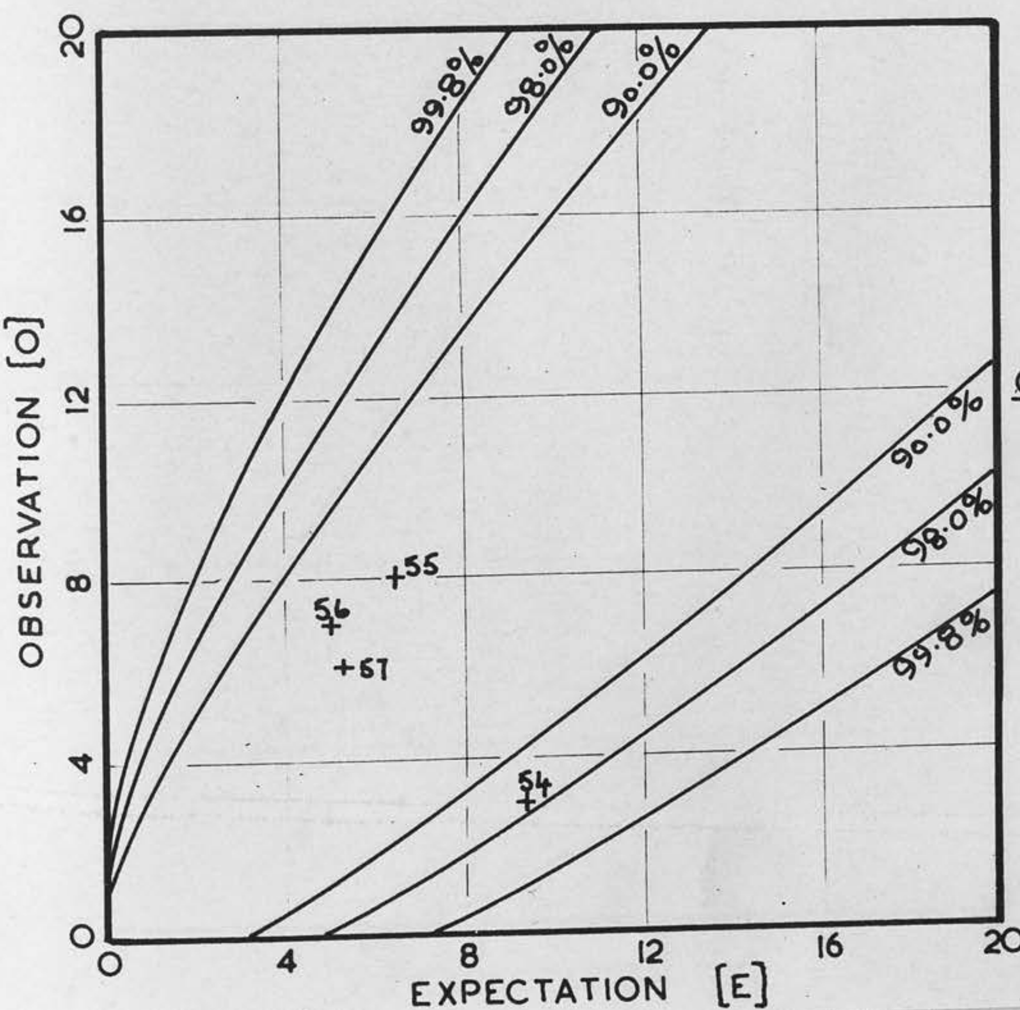
GRAPH XX

LOTHIANS AREA



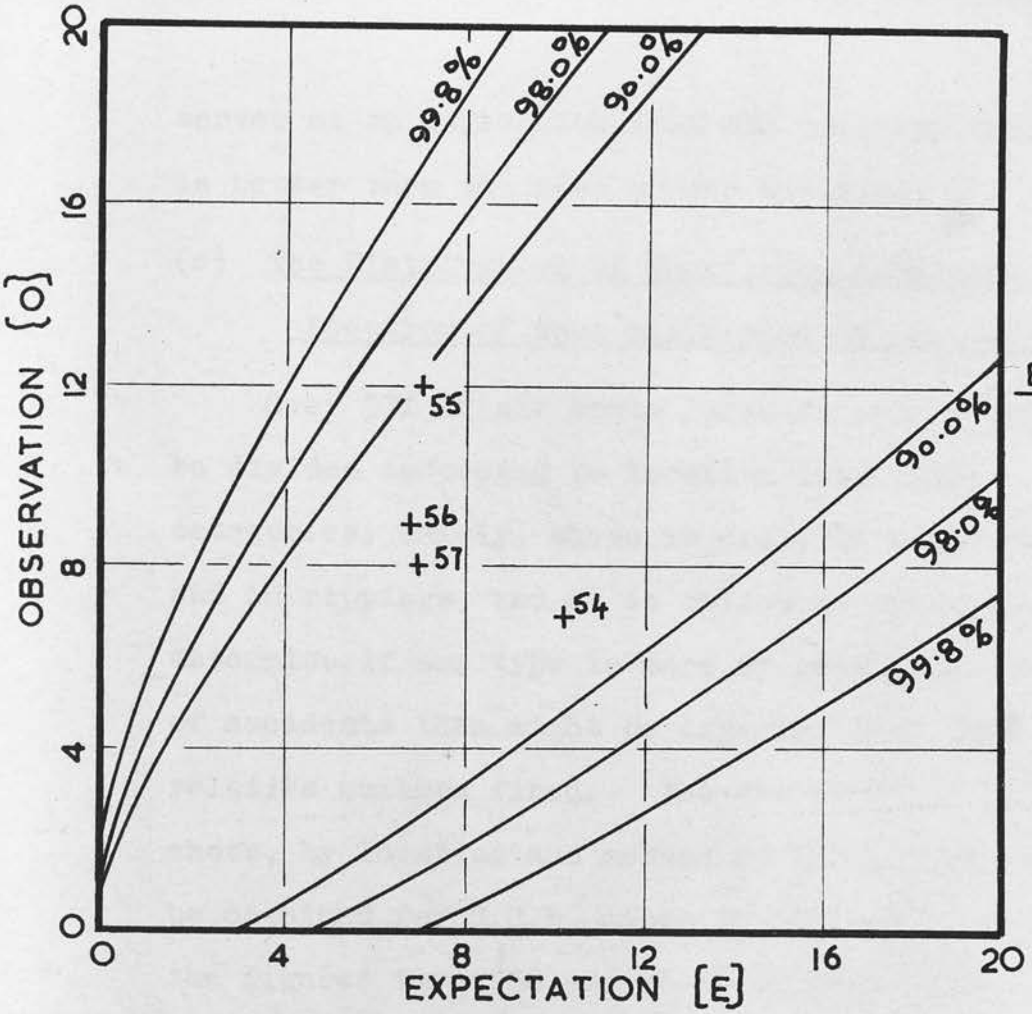
GRAPH XXI

EAST FIFE AREA



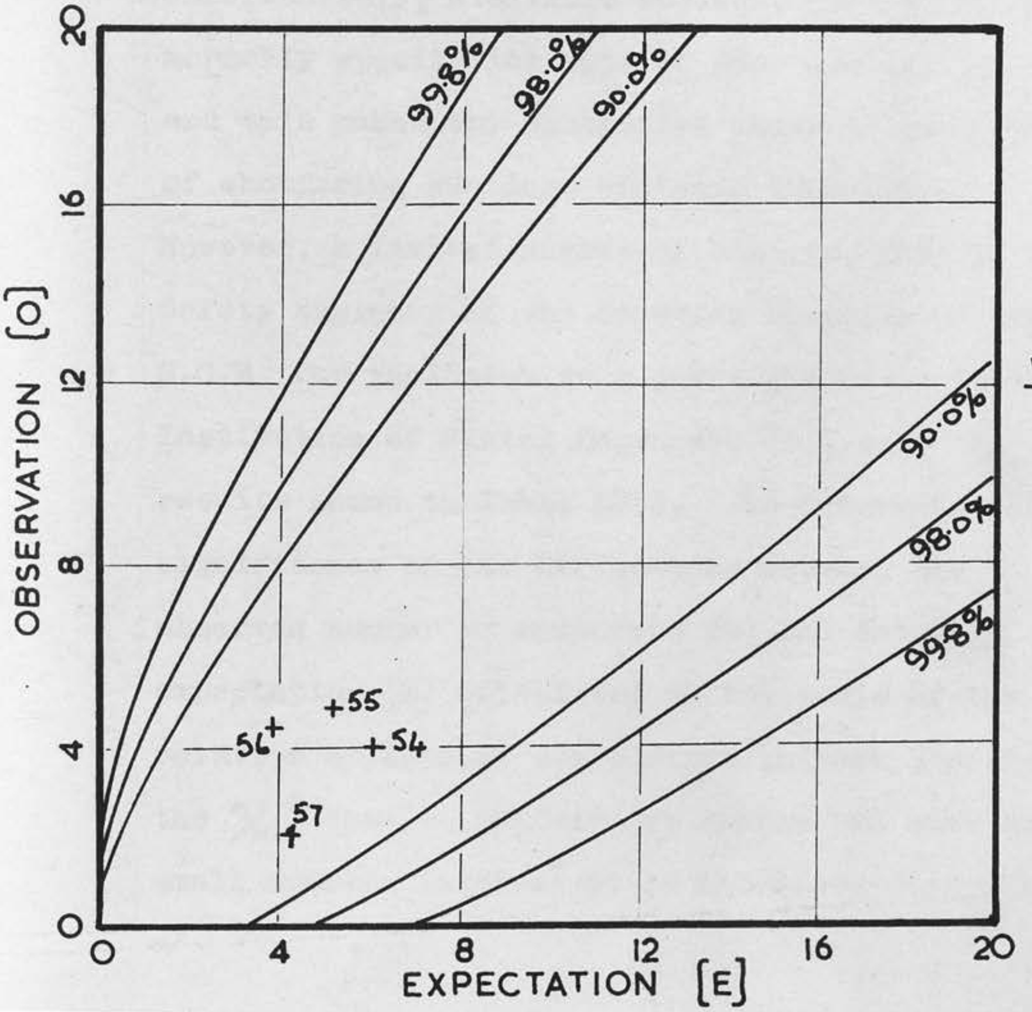
GRAPH XXII

CENTRAL WEST AREA



GRAPH XXIII

EAST AYR AREA



GRAPH XXIV

WEST AYR AREA

serves as an indication that the position there is better than the rest of the division.

(c) The Distribution of Shotfiring Accidents by Location of Shot and Method of Initiation.

Over 97% of all shots fired in collieries may be divided according to location into three categories, namely, shots in coal, in stone mines and in rippings, and it is obviously important to determine if any type is more or less productive of accidents than might be expected from the relative numbers fired. The distribution of shots, by location and method of initiation, may be obtained for N.C.B. mines for recent years and the figures for 1958 are shown in Table XVI. Unfortunately, available accident reports do not normally specify the type of shot causing injury and this makes any exhaustive study of this part of shotfiring accident analysis impossible. However, a limited number of results, kept by the Safety Engineer of the Scottish Division of the N.C.B. for inclusion in a paper presented to the Institution of Mining Engineers (v), gave the results shown in Table XVII. To determine the significance of the differences between the observed number of accidents (O) and the expectation (E) calculated on the basis of the relative numbers of shots fired in each location, the χ^2 test is employed as before but with the small numbers involved it is necessary to apply

a/

TABLE XVI

The Distribution of Shots Fired in N.C.B. Mines in 1958, by Location and Method of Initiation

Method of Initiation	LOCATION								Totals	
	In Coal Seams		In Stone Mines		In Rippings		In Other Locations			
	A	B	A	B	A	B	A	B	A	B
Single Shot	49.770	53.0	0.689	0.7	9.549	10.0	1.078	1.2	61.086	65.0
Simultaneous in rounds of up to six shots	14.551	15.5	1.769	1.9	4.239	4.7	0.772	0.8	21.431	22.8
Delay	0.301	0.3	3.152	3.4	0.286	0.3	0.361	0.4	4.100	4.4
Fuse	1.243	1.3	0.003	0.0	0.369	0.4	0.091	0.1	1.706	1.8
Pulsed Infusion	1.855	2.0	0.0	0.0	0.0	0.0	0.0	0.0	1.855	2.0
Alternatives to Explosives	3.290	3.5	0.001	0.0	0.191	0.2	0.278	0.3	3.760	4.0
Totals	71.010	75.6	5.614	6.0	14.734	15.6	2.580	2.8	93.938	100.0

NOTE For each Location, column A gives the number of shots in millions, and column B this number as a percentage of the grand total of 93.938 millions.

TABLE XVII

The Distribution of Shotfiring Accidents by Location of Shot - Scottish Division, 1952

Location of Shot	Shots fired		Observed Accidents (O)	Expected Number of Accidents ($E = P \times 63$)
	Number in millions	Number as percentage (P) of total		
In Coal Seams	10.133	73.0	40	46.0
In Stone Mines	1.114	8.0	2	5.0
In Other Locations	2.640	19.0	21	12.0
Totals	13.887	100.0	63	63

NOTE Prior to 1956, shots were divided by location only into the three categories shown, with 'Other Locations' including rippings. As shown in Table XVI, rippings account for over 80% of the shots fired in these locations, and roadway repairs etc. for the remaining 20%.

a correction due to Yates (vi) which reduces the difference between the observed and expected numbers arithmetically by a half.

$$\underline{\text{i.e.}}\ \chi^2 = \frac{\left[(O - E) \pm \frac{1}{2}\right]^2}{E}$$

the positive sign applying when $E > 0$, and the negative sign applying when $E < 0$.

Evaluating χ^2 for the three locations we find:

$$\chi^2 \text{ coal seam} = \frac{5.5^2}{46} = 0.66$$

$$\chi^2 \text{ stone mines} = \frac{2.5^2}{5} = 1.25$$

$$\chi^2 \text{ other locations} = \frac{8.5^2}{12} = 6.00$$

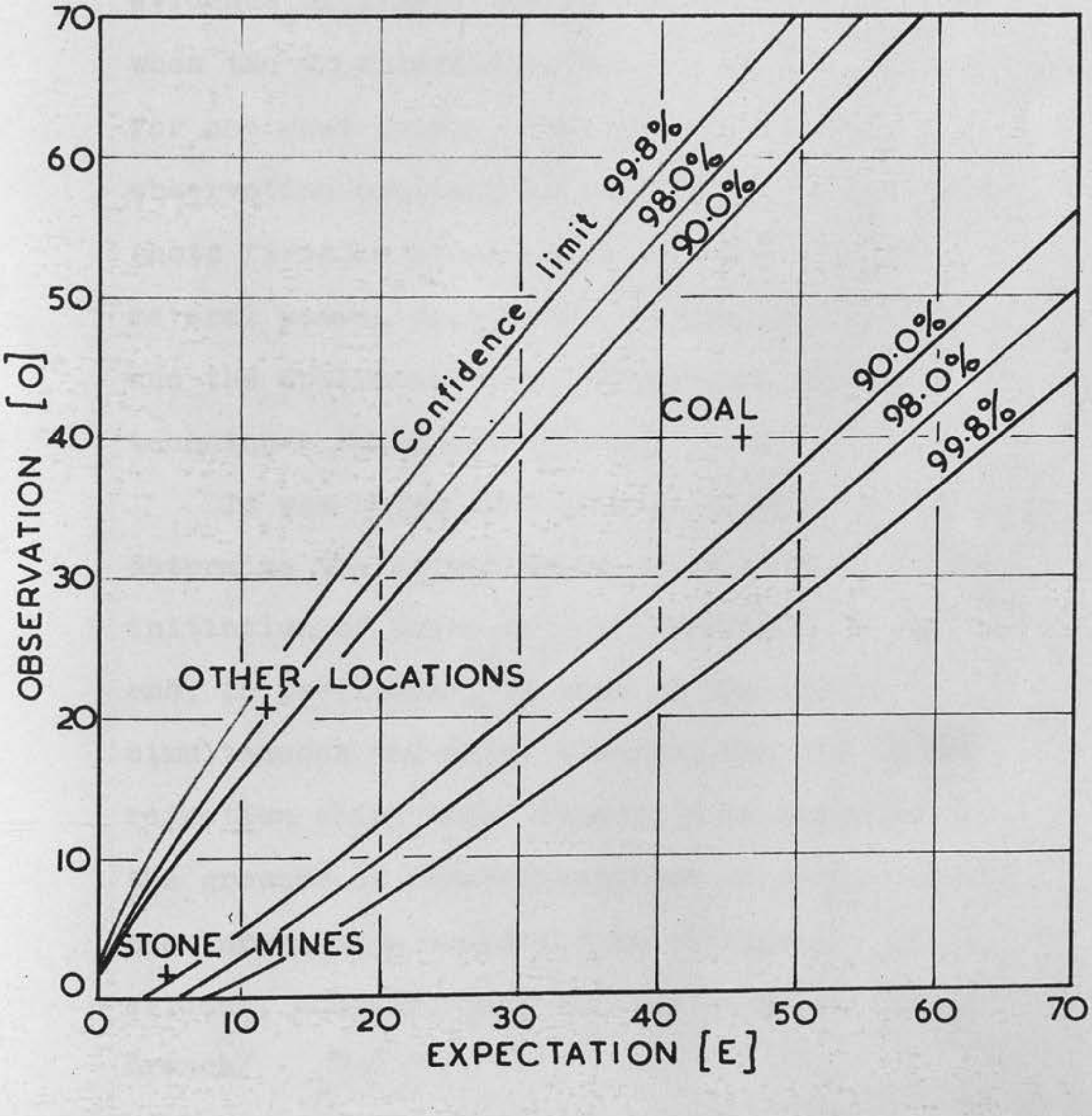
Using the same terminology and standards as in the earlier part of the Chapter, we see that the values of χ^2 obtained for shots in coal seams and stone mines show that no difference, unexplainable by random fluctuation, exists between the observed and expected values. However the figure of 6.00 calculated for χ^2 for shots in 'other locations' indicates that this type of shot is significantly more productive of accidents than the others at the 98% confidence level. This conclusion may also be derived graphically by plotting observation against expectation for each type of shot and this is done on Graph XXVI.

The probable reason for this disparity is that these shots, mainly in rippings, are so placed/

GRAPH XXVI

THE OBSERVED NUMBER OF ACCIDENTS [O] RESULTING FROM SHOTS IN COAL, STONE MINES, AND OTHER LOCATIONS, PLOTTED AGAINST THE CORRESPONDING EXPECTATION [E]

SCOTTISH DIVISION 1952



placed that any debris projected may travel long distances without meeting any obstruction in places where several men may be working. Only effective shelter can ensure freedom from accidents in these circumstances and where this is not provided or not used, the firing of these shots presents obvious hazard.

Two important points must be made when considering these results. The figures, referring to the Scottish Division, relate to approximately 15% of the national total and the value of conclusions drawn must be correspondingly limited. Also it is almost impossible to obtain positive evidence of significantly low accident figures when the expectation is 5, when considering results for one year only. However, should the observation continue at the level at 2, as with shots fired in stone mines in Scotland, for several years, this would then become significant and the application of further statistical techniques would reveal this fact.

It was hoped that some work could be done to determine the significance of the method of initiation of shots on the occurrence of accidents and, in particular, to show if the use of simultaneous and delay blasting had led to the reduction which might logically be expected on the grounds of reduced exposure to risk. Again, the information could not be obtained. It is evident, however, that further study of this

branch/

branch of shotfiring accident analysis might, with the co-operation of the interested parties, give valuable results and pointers for the positive work of accident prevention.

Conclusions

(1) The statistical techniques used in this work developed originally by Wynn (vii) provide a ready and valuable means of demonstrating graphically the significance of differences between observed numbers of accidents and corresponding calculated expectations.

(2) The accident rate in shotfiring operations has fluctuated violently over the period 1934-56, and is now very significantly lower than the 23-year average.

(3) The geographic distribution of shotfiring accidents is comparatively uniform. Only the Scottish and North Eastern Divisions show significant variations from the average for the rest of the country, with the position in Scotland being very bad and in the North East very good.

In the Scottish Division no area is significantly bad but there is some indication that the position in the Central East Area is better than the rest of the Division.

(4) On the very limited information available, it may be tentatively suggested that the firing of shots in rippings presents a greater hazard than shots in coal seams or stone mines.

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CHAPTER V

SHOTFIRING IN COAL

Introduction

Many variations, generally the result of local conditions and traditions, govern the method of work and, indirectly, the firing of shots on orthodox hand filled longwall faces, but normally the work of coal getting is cyclic and follows the pattern:-

Coal production shift	-	coal stripping, shotfiring
First preparatory shift	-	pack building, conveyor shifting, ripping
Second preparatory shift	-	coal cutting

Shotholes may be bored on any shift and in each case certain advantages and disadvantages accrue. On the coal filling shift the need for holes can be most accurately assessed but the addition of a hole borer, machine and cable to the complication already existing may cause confusion. On the preparatory shifts the concentration of men is lower but in most seams there is a tendency for the front of the coal to sag after the cutter has passed. This may result in holes bored on these shifts being wholly or partially cut off. It has been suggested, notably by the committee on the precautions necessary to ensure safety in the use of explosives in coal mines, that it would be desirable to have in large collieries a superior official who would be in charge of shotfiring practice, and would thus as part of his duties ensure that a suitable drilling pattern was adopted and used. Changing conditions underground/

underground would render a fixed pattern unsuitable but the constant supervision of this important operation in shotfiring is desirable, and even weekly inspection would ensure that indiscriminate boring is not taking place.

(a) <u>Shotfiring in coal with conventional methods and explosives</u>

As indicated in Table XVI over 75% of all shots fired in N.C.B. mines are in coal and of these over 90% are initiated by electric detonators fired singly or simultaneously in rounds of up to six shots. It is immediately apparent that together these classes form the most important subdivision of shotfiring, and they are therefore considered in the greatest detail.

When firing on a longwall face where permitted explosives only may be used, three methods may be employed, namely, single shot firing, single shots in groups of up to six and rounds of up to six shots simultaneously. The differences in procedure are slight and concern only the precautionary examinations and, for simultaneous firing, the nature of the detonator leads, cable and exploder used. Although the content and meaning of the Coal Mines (Explosives) Regulations, 1956, governing the work of shotfiring, are quite clear a discussion of the significance of the requirements forms the next section of this Chapter, as no previous record of such an analysis /

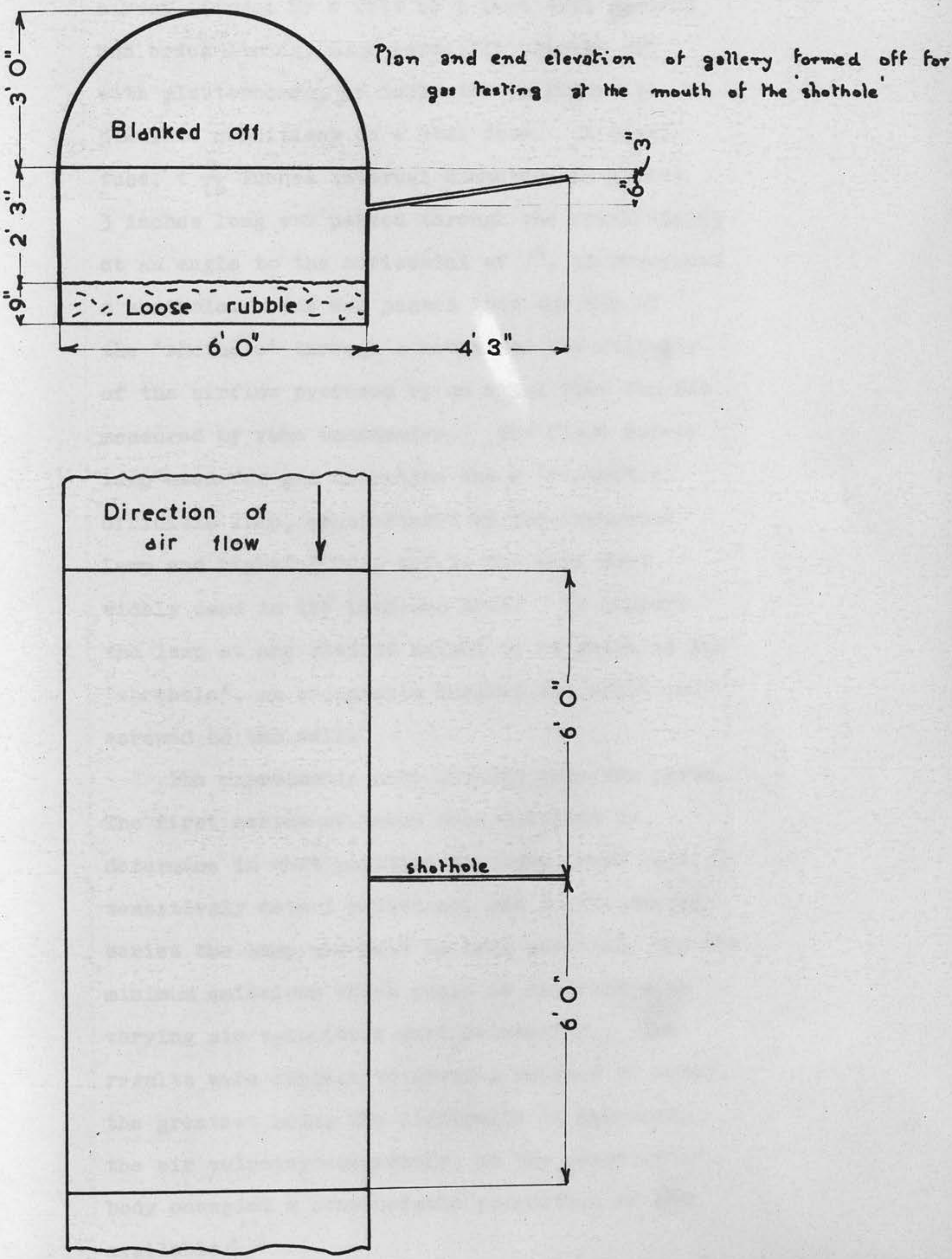
analysis could be found. Each part of the procedure is examined and the provisions reviewed in the light of recent work on the subject. The shotfirer must:

(1) Immediately before charging a hole test for gas at the mouth of the hole and all accessible places within ten yards, or twenty yards if the hole is within that distance of a roadhead on a longwall face. For group or simultaneous firing the examination is extended to include all the holes in the group or round, the area between the extreme holes and the waste, and all accessible places within ten yards of the extreme holes. 43 (5) b, (6) a. (The figures and letters refer to the relevant paragraphs and sections of the Regulations).

It is certain that this part of the work of charging and firing shots is invariably skimped and frequently omitted. This must be condemned wholeheartedly, as investigations (i) have shown that is is almost always the existence of a considerable body of methane/air mixture, detectable with good standards of practice, which makes an ignition resulting from shotfiring evident and that the remedy for such occurrences lies largely in the shotfirers' hands.

To determine the significance of that part of the test requiring examination at the mouth of the shotholes, a short series of experiments were carried out in a gallery used for mine rescue trainings/

FIGURE 11


Plan and end elevation of gallery formed off for gas testing at the mouth of the 'shothole'
Blanked off
Loose rubble
3' 0"
2' 3"
9"
6' 0"
4' 3"
6"
3"
Direction of air flow
shothole
6' 0"
6' 0"

trainings at the Heriot-Watt College. The airway, formed by 6 feet by 6 feet arch girders and brick lining, was partially blanked off with plasterboard, as indicated in Figure 11, to simulate conditions on a coal face. A steel tube, 1 $\frac{9}{16}$ inches internal diameter and 4 feet 3 inches long was passed through the brick lining at an angle to the horizontal of 7^o, to represent a shothole. Gas was passed into the top of the 'shothole' through a meter and the velocity of the airflow produced by an axial flow fan was measured by vane anemometer. The flame safety lamp used for gas detection was a 'Protector' officials lamp, manufactured by the Protector Lamp and Lighting Co., and is the type most widely used in the Lothians Area. To support the lamp at any desired height in relation to the 'shothole', an adjustable bracket was built and screwed to the wall.

The experiments were divided into two parts. The first series of tests were designed to determine in what position the lamp could most sensitively detect emissions, and in the second series the lamp was held in that position, and the minimum emissions which could be detected with varying air velocities were determined. The results were subject to several sources of error, the greatest being the difficulty in measuring the air velocity accurately, as the observer's body occupied a considerable proportion of the available/

available air space. It was also found that comparatively small obstructions, such as a hand, could increase or decrease the gas cap ob-:served. The effect produced depended on whether the air flow was restricted in the neighbourhood of the lamp, causing an increase, or channelled to flow past the lamp, causing a reduction. For those reasons it was extremely difficult to obtain reliable or repeatable results but the order of the emissions which are required before detection can take place make those reservations unimportant when considered in terms of practical applications. Certain main conclusions can, however, be drawn.

(a) As might be expected, the greatest sensitivity is achieved when the gas ports of the lamp are opposite the top of the shothole.

(b) For any given air velocity, a small change in emission from the shothole is sufficient to bring about a change in the lamp from a non-detectable cap to burning in the gauzes. In practice, the experimental variations referred to above completely overshadowed any variations caused by changes in the gas flow.

(c) The minimum detectable emission increased with increasing air velocity. Due to experimental inaccuracies and the limit on the maximum gas flow imposed by the apparatus, it was impossible to determine from the results the law/

TABLE XVIII

The gas cap produced by varying the position of the lamp in relation to the shothole, with a constant gas flow of 6.1 litres per minute

Position of lamp ports in relation to top of shothole	Gas caps, expressed as percentages of methane, produced in air velocities of 30'/min.	55'/min.	105'/min.
Ports above top of hole by 3/4"	3 - 3½	1½	0
" " " " " " 5/8"	3 - 3½	2	0
" " " " " " 1/2"	3½ - 4½	2	0
" " " " " " 3/8"	4½ - Gas burning in gauzes	2½	0
" " " " " " 1/4"	Gas burning in gauzes	2½	0
" " " " " " 1/8"	"	2½ - 3	Trace
Ports level with top of hole	"	3	1½
Ports below top of hole by 1/8"	"	2	0
" " " " " " 1/4"	3 - Gas burning in gauzes	1½	0
" " " " " " 3/8"	3 - 4½	1½	0
" " " " " " 1/2"	2½ - 2	0	0
" " " " " " 5/8"	2	0	0

TABLE XIX

The minimum gas flows required to produce measurable gas cap ($1\frac{1}{2}\%$) with the lamp supported with ports opposite top of the shothole, in different air velocities.

Air Velocity in feet per minute	Required gas flow in litres/minute
30	1.9
55	2.5
70	5.0
105	6.1
120	Above the maximum which could be supplied.

law governing the variation.

In practice (ii) where gas flows from shotholes have been measured very low readings have been recorded, ranging from 0 to 150 mls. / min. for holes in solid coal to negligible quantities from holes in rippings. The minimum gas flows detected in the tests was very much greater than this, and where ventilation comparable with normal longwall standards is passing, very large gas flows would be required before detection with a flame safety lamp could be achieved. The difference in gas flows from holes in solid coal and rippings is especially significant, as the probability of ignition is approximately ten times higher with ripping than coal shots (ii). It would thus appear that that part of the test requiring examination at the mouth of shotholes could well be omitted and the emphasis shifted on to a careful examination in all accessible places near the shothole.

(2) <u>Clean out shothole</u> and test for breaks. 24 (1) 45 (2). Every shotfirer in a mine in any part of which the use of lights other than permitted lights is unlawful, or where safety lamps are being used as a temporary precaution, must be provided with a scraper and break detector of approved design. These are combined to form one tool.

Shotholes generally retain a considerable quantity/

quantity of coal dust from boring, and where this is introduced between cartridges the risk of non-detonation of part or all of the main charge is increased.

The break detector is essentially a metal prong, not more than $\frac{3}{32}$ in. wide or thick at the tip, attached to a rod or tube, and is designed to be capable of detecting any break of $\frac{1}{8}$" or over crossing the hole either longitudinally or transversely. No hole containing such a break may be charged.

It is interesting to examine this legal requirement in the light of investigations carried out by the North Eastern Divisional Shotfiring Committee, detailed in their Third Report on Shotfiring in Yorkshire - The Ignition Hazard due to Shotfiring (iii). Tests carried out to determine the incidence and magnitude of breaks in undercut coal showed that out of 436 holes examined in the Top Hard seam only 40 were completely free from breaks and 247 had breaks which would cross the explosive charge. Many of the breaks were $\frac{1}{4}$" or more wide and could be identified in several holes. In many cases the firing of a shot resulted in the creation of new breaks or the enlargement of existing breaks in nearby holes. Roof conditions affected the total number of breaks, but even in exceptionally good conditions the percentage of break free holes was not increased/

increased. Similar tests in other seams showed that the incidence of breaks was dependent on the nature of the coal, but comparable frequencies were obtained confirming the results above. Tests in solid coal seldom revealed breaks and those which did occur were almost invariably in the front of the coal.

Gas analysis of samples taken from holes before shotfiring showed that 70% contained methane/air mixtures above the upper inflammable limit, and less than 10% within the limits. Samples taken from a hole before and after shotfiring in an adjacent hole revealed a reduction in methane content and thus a tendency for the atmosphere in the hole to be brought within the explosive limits.

It is evident from these results that the phenomenon of visible ignitions resulting from the use of explosives must be due to factors other than the occurrence of breaks of ¼" width containing methane/air mixtures within the inflammable limits crossing the charge, as these primary requirements are apparently satisfied in a comparatively large number of shotholes. An investigation in 1943 (i) of several ignitions of methane following shotfiring revealed that breaks of considerable width had crossed the explosive charge, but also that only because considerable bodies of methane were present were the ignitions noticed. In most cases this could have/

have been detected on careful examination with a flame safety lamp and the need for careful and comprehensive precharging and prefiring gas tests is thus emphasised.

(3) Compare the depth of the shothole(s) with the undercut or shear. 44. In situations where permitted explosives only may be used, shots in coal must have an additional free face provided, and the depth of this must be greater than the depth of the shothole. This requirement, which does not apply to anthracite mines or cross measure drifts crossing a coal seam, ensures that efficient use is made of the explosive and that the possibility of a blown out shot is minimised. The ignition risk attendant on a shot blowing out is very small, as it is under these conditions that an explosive is tested for official approval. However, holes in undercut coal require approximately one-fifth the charge necessary for shots with a comparable burden in solid coal and a considerable reduction in explosives consumption is thus achieved.

Where coal cutters are fitted with an efficient gummer, the cleaning of the cut is achieved mechanically but the operation of hand gumming is difficult, requiring in many cases the use of a special shovel. There is naturally a tendency for this work to be skimped, should the shotfirer not insist on it being carefully carried out.

(4)/

(4) Seal the back of the hole(s) with plugs of stemming. 48 a.

Following the discovery (i) that shots fired partially confined constituted a dangerous ignition hazard, the principle of testing for breaks was introduced. Sealing the back of the hole provides a protection for a break undetectable by the normal approved detector.

(5) Charge the shothole. (a) Ensure that there is no charged shothole within 30 yards of the hole about to be charged 25 (3). This provision does not apply to simultaneous firing. By experience it has been found that confusion with cables may result in the wrong shot being fired when shotfirers are working close together, and the distance of 30 yards chosen allows a reasonable safety margin without causing undue interference to the normal working in a section.

(b) Ensure that the hole is not overcharged. 26 (5). The amount of explosive required for any shot is dependent on so many factors, including the nature of the coal, the burden of the shot and the number of free faces present, that literally only after a shot has been fired is it possible to say definitely whether it has been correctly charged. Certainly, the number of holes wasted through undercharging is small in comparison with those which are overcharged, but the heavy manual labour involved in picking down coal not dislodged by a shot not unnaturally/

unnaturally creates a tendency for more explosive to be used than is strictly necessary.

(c) Ensure that only complete cartridges of the same type and diameter are used, and that these have $\frac{1}{8}$" diametral clearance in the shothole. It is essential that enough cartridges of small weight are included in the supply to the face to remove any temptation from the shotfirer to split cartridges. When faced with the alternative of breaking the law by either overcharging a hole, or cutting a cartridge, a shotfirer invariably chooses the latter course, and this may lead to the jettisoning of explosives underground.

(d) Ensure that the explosive is permitted, where this is required 2 (1), (2), (3), a, b, c, (4). No chance of confusion exists in most collieries, as only permitted explosives may be taken below ground in a mine in any part of which the use of permitted lights is required. However, in naked light mines permitted explosives must be used when safety lamps are being used as a temporary precaution, in intake airways, in or within 30 feet of haulage road or in a place designated dry and dusty by a Mines Inspector, and a real possibility exists of non-permitted explosives being used. A close control must be kept on the issue and transport of explosives to these situations.

(6) Prepare the primer cartridge. This is done by piercing the cartridge with a non-ferrous/

ferrous pricker, inserting the detonator, and hitching the leading wires round the cartridge to prevent withdrawal of the detonator on charging.

(7) Stem the shothole with sufficient suitable non-inflammable material 28(i). Experiment has shown (iv) that the most suitable stemming material is moist sand with sufficient clay added to render the mixture easily shaped into plugs for insertion into a shothole. In practice, a large variety of materials are used and not infrequently the stone dust provided in the gates is mixed into clay with water and formed into pellets. When a large number of shots are fired in a shift, the temptation to use readily available but undesirable coal dust is increased, and only a sense of responsibility in the shotfirer, coupled with supervision from higher officials, can eliminate the practice.

(8) Determine the danger zone likely to be created by the firing of the shot(s) 35(1), and post and instruct sentries or erect fences on all approaches to the shot(s). Should an excess quantity of explosive be used, or should the shot blow out, material will be projected often with considerable violence over great distances, and the determination of the danger zone must take this possibility into account. It is tempting and trite to say that the occurrence of an accident reveals a failure on the part of the shotfirer/

shotfirer to fulfill the requirement of this section. However, this is an extremely difficult problem, dealt with in greater detail later in this work.

(9) Make a prefiring examination for gas at each shothole and at the edge of any waste opposite 43 (6) b. This is necessary to ensure that no dangerous emissions of firedamp have occurred in the interval between the precharging tests and the prefiring test.

(10) Prepare to fire the shot(s).

(a) Connect the detonator leads to a cable of the correct type and minimum length 37 (3). No minimum distance from the shot is specified, but with single shot firing a cable of not less than sixty feet must be used, and for simultaneous firing in coal the corresponding length is one hundred and fifty feet. With simultaneous firing the detonator leads must be copper 27 (3) b, the detonators must be connected in series and no additional wire may be used to connect the leads to each other or the cable 32 (2) a, b. This is necessary to ensure that the resistance of the external circuit does not become so high that missfires would result.

(b) Ensure that the cable is kept free and clear of all electrical apparatus, including cables 32 (4). Low tension detonators are very sensitive and comparatively small applied voltages may cause detonation of the main charge.

(c)/

(c) When simultaneous firing is being employed, test the circuit for continuity 32 (7). All approved six-shot exploders incorporate a continuity test and the use of this at once indicates an open circuit in the external wiring, a possibility which is much greater with the larger number of connections in multi-shot firing. The continuity test is also indispensable for isolating a detonator with a broken internal connection.

(d) Fire shot(s) after ensuring that everyone is in shelter.

(11) Make an after firing examination for gas and general safety, recall and instruct workmen on any steps to be taken 37 (1) a, b. This requirement, which imposes on the shotfirer an obligation to return to the scene of the shot(s) before the workmen, was first introduced in the Explosives in Coal Mines Order (1951) and is designed to ensure that he, with his presumably greater experience, will guide the men on any steps which may have to be taken to make the place secure.

(12) With group firing, proceed at once to charge the next hole in the group.

Fuse firing, when permitted, is exempt from many provisions of the Regulations, and all the requirements designed to prevent ignitions of methane are omitted. Certain articles, inapplicable/

inapplicable to electric shotfiring, are introduced and these relate to the minimum lengths of fuses for single shots and rounds, (three and four feet respectively) the allowable proximity of naked lights to explosives and detonators,(four feet and not directly above) and the minimum separation of charged shotholes when firing single shots (50 yards). Fuse firing is now of little importance, and only in the Scottish Division is it practised to any appreciable extent.

The comparative importance of single shot, group and simultaneous firing may be judged by reference to Table XVI, which shows that single shot and group firing account for 70%, and simultaneous firing 20%, of all shots in coal. Simultaneous firing is not altogether suitable for orthodox longwall working, as the instantaneous detonation of up to six shots would result in a large amount of roof being exposed. Because of the comparatively slow speed of hand filling, this would be unsupported for a considerable time. In fact, this method has found its greatest application on power loading installations, where the machine employed requires some preparation of the coal by explosives. In these situations it is important to note that a team of men is available to set supports immediately the coal has been loaded/

loaded out, and that the delay with single shot firing would result in an expensive machine not being used to its full capacity.

No differentiation between single shot and group firing can be made but it is interesting to note that the Regulations place an obligation on the shotfirer to employ group firing 'if shots are to be fired singly in succession in coal along a longwall face' (v). The difference in practice is slight, concerning only the precautionary examinations, and it is in any case doubtful if the average practising shotfirer could identify his method with either of the alternatives allowed in the Regulations.

(b) Shotfiring in Coal with Alternatives to Explosives

When the dangers inherent on firing explosives were realised, attempts were made to alleviate the hazard by introducing devices which could perform the task of explosives without the attendant risk. Many of these quickly became curiosities because of their impracticable nature, but recently interest in this branch of blasting technology has been revived in the drive to minimise the dangers and increase the efficiency of shotfiring operations. The term 'alternatives to explosives' although not strictly grammatically correct, has been accepted as referring mainly to steel tube blasting devices, designed to provide the useful feature of explosives/

explosives - large volumes of gases at high pressure - without the attendant disadvantages - the initial shock wave on detonation and the high temperature of the gases produced. Reference to Table XVII shows that in 1958, 4% of the annual total of shots were fired with these devices, mainly in coal seams.

The three common systems in this country employ a steel tube in which the pressure is caused to rise to a predetermined value, when an expendable plate, pin or plug shears, allowing the gas produced to flow into the shothole. Extensive literature is available giving full details of the technical features of the shells, and only a brief description is given here. In the Cardox system the charge of liquid carbon dioxide is raised above its critical temperature by a heater unit, initiated by a normal approved exploder, and the rupturing of a shear disc allows the escape of the carbon dioxide under high pressure into the shothole. The action of the Hydrox shell is similar but the charge is a mixture of solids which react under influence of heat and pressure to evolve a large volume of gases. In the most recent type, the charge, heater and shear plug are incorporated in a composite unit, allowing reloading of the shells to be carried out at the face. The Airbreaker system employs compressed air as the blasting medium, and requires a connection to be maintained between/

between the shell and the compressor. Unlike the other systems, the discharge of the shell is controlled mechanically by a firing valve.

Alternatives to explosives possess advantages and disadvantages when compared with conventional explosives and each other, and it is this balance which controls the introduction of these devices in any particular set of circumstances. Thus the advantages may be summarised as follows:-

1. Freedom from the dangers of ignition of methane/air mixtures.

As already pointed out in Chapter II, any device operating at over 450 pounds per square inch can give rise to ignitions in methane/air mixtures by causing adiabatic compression in the mixture. It is, however, extremely doubtful if the necessary close confinement could be achieved underground, as the first movement of the material being blasted would afford a partial release of pressure and will thus prevent the necessary temperature rise. Certainly no ignitions have been recorded with these devices underground, and their high margin of safety in the presence of methane/air mixtures is recognised in the Coal Mines (Cardox and Hydrox) Regulations, 1956, which impose much less strict requirements than the corresponding Explosives Regulations. In particular, their use is permitted in the roof of longwall workings between the coal face and the waste, in which situation conventional explosive/

explosive is completely forbidden.

The risk of ignitions is not entirely eliminated by the use of alternatives to explosives, as Cardox and Hydrox require conventional exploders which may become dangerous if defective, and Airbreaker compressors sited underground may overheat should protective switchgear become defective. Experience has shown that these possibilities are more apparent than real, and it was undoubtedly the high safety margin of these alternatives to explosives which originally led to their introduction in situations where high methane emissions might be expected. Recently, however, economic reasons have brought about their introduction in comparatively risk free environments.

2. Improvement in the size analysis of the coal product.

All conventional permitted explosives are of the detonating type, and the production of gas is accompanied by a shock wave on detonation. This property is of great value when hard materials are being blasted and the velocity of detonation of an explosive is one of the controlling factors influencing its ability to shatter strong rocks. In coal blasting work, however, this property is not required or even desirable and leads to degradation and consequent reduction in value of the product.

Alternatives/

Alternatives to explosives produce the gases necessary for blasting without the initial shock wave, and the same energy is released over a much larger time interval than is the case with conventional explosives. They thus exert a comparatively mild action on the material being blasted and are not suitable for blasting hard or very fissured rocks. The shattering effect of conventional explosives may be reduced by lowering the velocity of detonation but cannot be eliminated altogether. Tests carried out in a joint investigation by the Institution of Mining Engineers and the National Coal Board, and reported in the Fourth Report of the Committee on Shotfiring and its Alternatives - A comparison of various methods of shotfiring (vi) showed that in the same seam, there was no significant difference in size analysis of the product when detonating explosives were used, whether the shots were fired singly, simultaneously, in rounds of short delays, with pulsed infusion shotfiring or orthodox single shots in coal wetted by infusion. For each method, the percentages of the oversizes were plotted against the size of the screen openings to which they related in accordance with the Rosin-Rammler method and a measure of the coarseness of the material obtained by noting the screen size corresponding to an oversize of 36.79%, as is usual with this method. Conventional explosives/

explosives gave a screen size of 3 inches, whereas Cardox (the alternative used in the tests) gave a corresponding size of 4 inches. Those results are very valuable, as they give one of the few records of tests carried out by the same people in the same situation, under closely controlled conditions, and are thus free from the temporary improvement which may be effected when any new method is first introduced. Although the tests refer only to Cardox blasting, there is no reason to suppose that the other alternatives would produce inferior results.

3. A high rate of firing may be maintained.

Because of the high margin of safety of these devices in the presence of methane/air mixtures, the procedure for firing is greatly simplified in comparison with conventional explosives. This reduces the time taken to fire a single shot to approximately eight minutes, as reference to Chapter VII, page 158, and the Coal Mines (Cardox and Hydrox) Regulations, 1956, shows that operations 1, 2, 4, 6 and 7 may be omitted. Where conditions are suitable, a face may be precharged with Cardox and Hydrox and a further saving in time would result.

When Airbreaker installations are used very high firing rates can be maintained, as reloading the shell involves only the replacement of/

of the expendable shear pin or disc, and the same shell is re-used for each shot. No legislation as yet directly affects the firing of Airbreaker shots, and until this is placed on a similar footing to the other alternatives any comparison in firing rates would be profitless.

4. Reducation in projected material.

As the shotfirer has only very limited control over the energy released on the firing of the shot with alternatives to explosives (page 116) it might be expected that their use in holes where the burden is small would result in an excessive amount of material being projected. In practice it appears that venting provided by the first movement of the coal or stone being blasted allows any excess energy to be dissipated harmlessly. With conventional explosives the available energy is released so rapidly any excess over actual requirements causes breakage of the coal and projection of any available coal or stemming. Unfortunately, due to the comparatively limited numbers of shots fired with these alternatives, and the limited accident information available this cannot be proved or disproved analytically but visual indications at an installation of an Armstrong Airbreaker visited, and the opinion of experienced mining engineers, do indicate a considerable/

considerable reduction in the hazard from projected material.

5. Reduction in fume and dust production. Although the fumes produced by modern permitted explosives are almost completely non-toxic, the use of alternatives eliminates any possibility of poisoning from this count.

As a corollary of increased large coal production with alternatives to explosives, dust production is generally reduced and this leads to an improvement in environmental conditions.

The disadvantages of alternatives to explosives may be summarised:

1. A very limited range of powers are available.

Although shells are manufactured in a considerable variety of diameters and volumes, the dimensions are normally standardised for a district in a colliery, and often for the complete colliery. Thus, the size of the Cardox and Hydrox shell must be decided on the most arduous duty likely to be encountered, and the use of this size in other situations implies economic disadvantages in employing a device too powerful for actual requirements. In Airbreaker installations a limited degree of control can be exercised by varying the thickness of the shearing pin or plate but, compared with faces using conventional explosives where a range of charges of 500% (3 oz. to 15 oz.) may regularly/

regularly be used, the control is very limited.

It would thus appear that alternatives to explosives would be under a great disadvantage under this head but, in practice, this is not noticeable.

2. Risk of projected shells.

Shells have forward facing gas ports or sprags operated by the escaping gases as a precaution to prevent the ejection of the shell from the hole. However, should reasonably rapid release of pressure not occur, the gases may build up sufficient pressure behind the shell to cause violent projection of the shell. Paragraph 18 (1) b of the Coal Mines (Cardox and Hydrox) Regulations, 1956, states that no one shall fire a shot unless 'measures of a kind specified by the Manager of the mine have been taken to ensure that the shell is not ejected in a dangerous manner on firing'. The consequences could be very serious indeed and much more damage could be caused by a large heavy shell than the comparatively limited amount of material projected by a shot with conventional explosive.

3. Inconvenience of 'fizzers'.

When the heater of a Cardox or Hydrox shell operates, but the pressure does not rise to the value necessary to shear the disc or plug, an interval of ten minutes must be observed before the shell is approached. These occurrences are/

are much more frequent than missfires with conventional explosives, and have a very high 'nuisance value'.

In Airbreaker installations, should the disc or nail fail to shear, the shell can be immediately vented to atmosphere, and the fault rectified safely without any waiting period.

4. Inflexibility.

Conventional explosives may be carried conveniently to any place in a colliery, but the use of alternatives generally requires fairly uniform working conditions to make the transport arrangements for shells economically feasible.

5. Maintenance.

All Cardox and the older type of Hydrox shells must be inspected at least once in 90 days, and this interval is reduced to 30 days for the new type Hydrox shell. This means that all shells must be numbered and checked regularly. Records of all such examinations must be kept.

6. Shothole drilling.

Alternatives to explosives generally require the use of large shotholes and this may create some difficulty in boring. In addition, as no stemming is used, holes are drilled to give little diametral clearance round the shell and in seams where the front of the coal tends to sag after cutting, holes must be drilled as soon as possible before blasting to ensure easy entry of the shell into the hole.

7./

7. Difficulties in handling underground.

In seams of limited height the handling of long heavy shells, accompanied in Airbreaker installations by lengths of hose, can present considerable practical difficulties, and it is likely that the minimum height for Airbreakers will be at least 2 ft. 9 in.

On occasion, the shell may be buried in the pile of blown debris and recovery may take some time.

8. Limitation in application.

Although the freedom from the shock wave on initiation is a very desirable property when coal is being blasted, it places limitations on the applicability of alternatives in other situations. This invariably means the employment of both alternatives and conventional explosives, with the accompanying disadvantages of duplicated records, storage facilities, etc.

As already mentioned, each alternative offers certain advantages and disadvantages in comparison with the other two types. Thus the Cardox system requires complicated charging plant, a pit supply of 300% of the normal days requirement of shells, and arrangements for the transport of shells underground. It is, however, the only alternative which may be used for simultaneous firing. The new Hydrox requires fewer shells as it may be recharged at the face, and is suitable for small scale trials as/

as no charging apparatus is required. Armstrong Airbreaker installations are comparatively inflexible, but where uniform working conditions exist, shells may be 'reloaded' and fired more quickly than the other types.

The study of the economics of alternatives to explosives in comparison with conventional explosives, or the comparison of one type with the other two, is a very involved subject and not within the scope of this work. So many variables must be taken into account, including capital charges, depreciation, maintenance, and intended utilisation that any assessment must be based on local factors. However, it appears that when outputs of 500 tons per day may be prepared by an Airbreaker installation, the cost expressed in pence per ton may be equal to conventional explosives, and any size increase in the coal product will result in economic advantages.

With the Hydrox system, the cost per shot remains approximately constant no matter how many shots are fired, as the main expense is the cost of the charge. The cost of preparing the coal will always be more than with conventional explosives. Cardox suffers from high transport costs, and again the cost expressed in pence per ton of output will be high. It must, however, be remembered that the introduction of alternatives may be dictated as a/

a precautionary measure in situations considered to present an unusually high methane hazard, and in these cases any economic disadvantages which may arise will not be considered so important.

(c) Shotfiring in coal with alternative methods.

1. Pulsed infusion shotfiring.

The use of water under pressure to infuse coal seams and thus assist in the suppression of dust is now a well known and widely applied technique, (viii), and since 1954 this has been combined with the use of explosives to prepare coal seams for hand or power loading. The technique, developed originally to reduce the possibility of ignition of methane by explosives, has also been claimed to offer advantages of increased large coal production, reduced explosives consumption, the elimination of the need for conventional stemming and the reduction of airborne dust produced on shotfiring.

The use of the method has been widely and fully described in technical literature (ix) and only a very brief description is included here. Submarine detonators and a special explosive capable of detonating satisfactorily under high pressures are required. In under:cut coal, the shothole is charged, an infusion gun sealed in the hole and the shot fired from a station, after it has been ensured from the readings on a combined flowmeter/pressure gauge that the water is flowing satisfactorily and that/

that the hole is offering adequate resistance. Only in exceptional cases does undercut coal offer enough resistance to a flow of water for this method to be applicable, as the operation of cutting normally induces breaks of consider-:able width to be formed in the coal seam. In solid coal, this technique may be used on longwall faces to prepare the coal for hand or power loading, or for longhole blasting in either level or highly inclined seams. The method has found its greatest application in the preparation of stable holes on power loading faces where the complication of a coal cutter is not justified, and where the high explosives consumption resulting from blasting off the solid is of little importance for the limited output won. Recently, millisecond delay blasting in conjunction with pulsed infusion shotfiring has been introduced, and offers the advantages of greater speed and reduced exposure to risk.

The advantages of this method are:-

(a) Freedom from the possibility of ignitions.

As early as 1872 mining explosives were employed in conjunction with water cartridges to reduce their high incendivity, and the Report of the Royal Commission on Accidents in Mines in 1886 concluded that all detonating explosives, if used in conjunction with water cartridges or porous, water soaked tamping were safe, if a shot blew out into an inflammable/

inflammable atmosphere. It is important to note that the tests employed imitated the conditions occurring when a shot blew out without doing any work, as this was thought to be the most dangerous situation possible. It is now held, however, that ignitions occur when an explosive charge is detonated in partial confinement, such as exists when a break in the strata crosses the charge, and the claimed safety advantages of pulsed infusion shotfiring must be considered in the light of this knowledge.

Over the period 1911 - 1957, only 71 ignitions resulted from shotfiring in coal seams have been recorded, compared with 130 following shotfiring in rippings (iii). When the comparative number of shots fired in coal seams and rippings is taken into account, it is found that the probability of ignition is ten times greater with ripping shots. As already pointed out (i) an investigation into many such ignitions revealed that in many cases the explosive charge was crossed by a break of considerable width, and it is in just those situations where pulsed infusion shotfiring cannot be applied. Even in undercut coal the occurrence of breaks usually precludes the use of the method and it is thus seen that only in environments already risk-free will this method be applicable.

It must also be mentioned that in a paper (x) presented to the Institution of Mining Engineers/

Engineers, Tideswell showed that over the period 1937-52 the operation of coal cutting caused almost as many ignitions as shotfiring, and thus presents a hazard in itself of several times that which it is designed to prevent. However, firing coal off the solid would greatly increase the amount of shotfiring necessary and the significance of this would be difficult to assess. Nevertheless, it does appear that too much stress should not be placed on the freedom from ignitions resulting from the use of pulsed infusion shotfiring, as its application is limited to situations in which dangerous conditions do not occur.

(b) Dust and fume suppression.

Experiments carried out in a joint investigation by the National Coal Board and the Institution of Mining Engineers (vi) showed that in four cases out of six examined, the total number of particles in the range 1 - 5 microns produced by firing 100 yds. of face in undercut coal by pulsed infusion shotfiring was less than that produced by orthodox single shot firing. Similar comparisons in a different location between pulsed infusion and orthodox single shots in coal wetted by normal infusion showed that in six out of seven cases pulsed infusion firing gave lower dust counts per 100 yds. of face fired. The reductions observed varied widely, ranging up to 82%. It would thus appear that the/

the use of this method does reduce the airborne dust created by shotfiring.

(c) Superior spreading action, resulting in reduced use of explosives and a greater percentage of round coal.

It is extremely difficult when analysing differences produced by changes in methods, to differentiate between short term improvements brought about by increased supervision of the new method, and any permanent benefits. Thus, while limited tests have showed spectacular improvements in large coal production, closely controlled size analysis of the coal produced by pulsed infusion shotfiring in undercut coal showed that in two cases an improvement in the +6", in one case no change, and in two cases a deterioration, compared with results obtained with conventional single shot firing (vi). It would thus appear that no significant difference exists in the size analysis of the coal produced by pulsed infusion and single shot firing.

In the same series of tests it was found that in four cases out of five the explosives consumption per unit of face length was reduced by amounts varying from 18 to 33%, and in the remaining case was increased by 14%. Should such savings be reproduced in large scale operations, considerable economic advantages would accrue.

(d) Stemming on tap.

Where a large number of shots are being fired/

fired, the preparation and transport of sufficient suitable stemming material can present problems, especially in districts with no local supply available. The use of water piped to the coal face removes any temptation from the shotfirer to use unsuitable materials.

The disadvantages of pulsed infusion shot-:firing, which have so far severely limited the extension of the method are:-

(a) Inconvenience.

The inconvenience of working with a hose and flowmeter/pressure gauge can be considerable, especially in thin seams. In addition, special arrangements must be made to ensure that only the correct explosive gets to the faces where pulsed infusion firing is in use.

(b) Applicability.

The great bulk of output from British coal mines is still won by hand filling of machine cut faces and only in exceptional cases can pulsed infusion shotfiring be applied in these circumstances. Reference to Table XVII shows that only 2% of all shots in 1958 were fired with this method, and it would appear that any further increase will be restricted to special applications such as the preparation of stable holes on power loading faces. Even then, the method cannot be used in very hard coals, as the gun may be ejected from the shotholes with considerable violence if the coal does not break and afford some/

some release of the high pressure impulse on detonation.

2. Off-shift firing.

As indicated in section (a) of this Chapter, the standard practice in this country is for the shots necessary to prepare the coal for hand loading to be fired on the production shift. In certain districts, however, some or all of the shots may be fired on the second preparatory shift, and this method, generally called off-shift firing, offers obvious potential advantages over the normal method. To gain some experience of the system two collieries in the Central East Area of the Scottish Division were visited for one week each, and the following impressions gained.

Colliery A was a small naked light mine approaching the end of its useful life and working mainly small areas of coal between faults. On the face visited the method of work was longwall advancing, but the total length of face was only 70 yds. No conveyors were used, as five roads were carried to the face, and the coal after being flung down the face was loaded directly into tubs which were then marshalled at the haulage road. The seam, three feet thick, was undercut to a depth of four feet six inches, and the gummer fitted to the machine effectively cleaned the cut to the back, giving three or four inches clear between the coal and the few remaining/

remaining cuttings. Support was by wooden bars and props and packs at each side of the roads.

The cycle of work was orthodox, being,

Production shift	- coal stripping
First preparatory shift	- ripping, pack building, hole boring.
Second preparatory shift	- coal cutting.

The cutter men were naturally finished early in the shift, and shotfiring did not start till they had left the face. As the shotfirer was normally the only man in the section, he erected notices in the approaches to the face. The actual procedure was not above criticism but in effect the shots were stemmed and fired in succession along the face, and the coal was effectively prepared for hand loading. The deputy on the production shift carried a few detonators, but there was rarely any need for additional shots.

At colliery B, a safety lamp mine, the method of work was orthodox longwall advancing in a six foot seam, with conventional conveying of the coal to the loader gate. The 100 yds. long face was undercut to a depth of four feet nine inches. Support was by steel props and bars, with packs and chocks at the waste edge. The cycle of operations was:

Production shift	- coal stripping.
First preparatory shift	- pack building, ripping, conveyor advancing.

Second/

Second preparatory shift - coal cutting, hole boring.

The shotfirer went on to the face halfway through the Second Preparatory Shift, and this gave the cutter men and hole borer time to get well clear before shotfiring started. The coal tended to heave in large blocks away from the face and required relatively little preparation with explosives, only 30 - 40 shots being fired over the complete length. The machine was not fitted with a gummer and the impression gained was that the shots were fired virtually on the solid. Nevertheless, effective results were obtained in the easy conditions. Again, the shotfiring operations were not above criticism but again it was difficult to visualise circumstances in which an accident could occur to anyone other than the shotfirer.

Advantages of off-shift firing.

1. Undoubtedly the main advantage is the reduction in risk of an accident occurring should material be projected from the shot or shots, as the number of men who could be injured is reduced to one or two, compared with up to ten or more when shotfiring takes place on the production shift. Obviously, for an accident to occur debris from a shot must hit a man, and as it is impossible in normal working to eliminate the projected material, it is desirable to reduce as far as possible the number of men that the/

the material can injure.

2. When shots are fired on the production shift, the time spent by facemen taking shelter is considerable and the constant repetition of this at short intervals can lead to carelessness. This danger is eliminated with off-shift firing.

3. The shotfirer is completely free to exercise his own judgment with regard to the charge to be employed. While it is true to say that this also applies on conventionally fired faces, it is likely that continued demands from the strippers for increased charges will have some effect.

4. Any fumes or dust created by shotfiring are dispersed harmlessly at a time when the least inconvenience is caused. Normally, few men are employed near the coal face on the second preparatory shift.

It was thought that at colliery A this was one of the main reasons for the use of off-shift firing, as the rather sluggish face ventilation did not clear the fumes from the shots and fuses for a considerable time after firing.

5. The shotfirer can work at a uniform speed throughout the shift. When the conventional methods of firing are adopted there is a marked tendency for shotfiring to be concentrated in short periods throughout the shift (Chapter VIII). This is inevitable as men working at approximately the same speed will always be ready for shots at approximately/

approximately the same time, and this can lead to great haste and skimped precautions. The use of off-shift firing does not, of course, guarantee that the provisions of the Regulations will be observed, but certainly removes at least one of the causes for such infringements.

Disadvantages of off-shift firing.

1. Working alone, normally without supervision, it is possible that the shotfirer may become careless and slipshod in his work. The influence of this factor could only be assessed after long experience with the method, but in the two collieries visited the standard of practice did not seem to be markedly different from that observed on conventionally fired faces.

2. After a shot has been fired, it is frequently difficult to tell to what extent the burden of the neighbouring hole has been relieved. This tends to lead to every hole bored being charged, even though this may not be strictly necessary.

3. Limitation of application because of roof control problems. It is obvious that this method can never be of universal application, as many roofs would not be capable of staying unsupported for the necessary period - approximately four to six hours. The importance of this is difficult to assess and only an actual trial would prove the feasibility of the method/

method in doubtful situations. At colliery B, the roof conditions were not good, and in fact a trial of a power loading machine working in conjunction with a prop-free front method of support had recently been abandoned because of the failure of the system to give effective roof control.

Should full face off-shift firing not be possible, the shotfirer could work across two shifts, firing half of the shots on the second preparatory shift and the remainder on the production shift. If more than one shotfirer is required the problem is eased, and cross-shift working would not be required.

If roof conditions proved difficult, an additional prop could be erected at the face end of the bars after the cutter had passed, and this would reduce the distance from the last row of props to the beginning of the solid coal by approximately two feet, or from six feet six to four feet six with a normal four feet six inches undercut.

3. <u>The use of conventional explosives in conjunction with non-inflammable plastic water filled bags (water ampoules).</u>

The idea of using water in conjunction with an explosive charge to obtain effective dust suppression of seams too hard to be infused by normal methods was developed by Demelenne in Belgium (xi). In his method, the amount of explosive/

explosive used was not sufficient to produce displacement of the coal, but recently interest in this country has been revived in this work, and the water is now incorporated in non-inflammable plastic bags, each holding approxi-:mately 250 mls. of water as part of the stemming in normal shotfiring operations. Many advantages have been claimed for this method and are detailed below.

(a) Reduced possibility of ignitions from shotfiring.

As already pointed out, the Royal Commission on Accidents in Mines in 1886 reported that all detonating explosives were safe if used in conjunction with a water cartridge or porous stemming, even if the shot blew out in the presence of methane/air mixtures, and it would seem reasonable that similar protection would be afforded by the use of water ampoules. Tests carried out in the explosives testing gallery at the S.M.R.E. (xii) with a non-permitted explosive, showed that the inclusion of 600 mls. of water in the cannon bore, raised the charge limit for inverse initiation from 8 to 24 ounces and thus effected a considerable increase in safety. The exact significance of any improve-:ment as measured in the blown out shot test is impossible to correlate to firing in breaks, as in the latter case the water is not interposed between/

between the explosive and the inflammable atmosphere. It would, however, appear that some increase in safety would result. In addition, this method may be applied in any situation, and is not limited, as is the case with pulsed infusion shotfiring, by the nature of the shothole.

(b) Reduced transport of stemming material.

The water ampoules may easily be carried in considerable numbers and filled with water at some suitable place near the face. In practice, it is rather difficult to make a watertight knot in the neck of the ampoule, especially in underground conditions, and this leads to a comparatively high loss of water on the face. Nevertheless, the reduction in the need to transport orthodox stemming may be a considerable advantage in certain districts where suitable material is not readily available.

(c) Reduction in dust produced on shotfiring.

As mentioned above, this method of using explosives in conjunction with limited volumes of water was developed to aid the suppression of airborne dust, and Demelenne recorded reductions of up to 40% in concentrations in coal winning operations subsequent to this method being applied. In this country, tests carried out by the N.C.B. Scientific Department (unpublished) show widely fluctuating results for shots in coal seams, but general agreement in the reduction in airborne dust/

dust observed with ripping shots, the average being about 40% and the maximum about 80%. Such startling reductions with the volumes of water used (200 to 300 mls.) are indeed remarkable and would seem to imply that the bulk of the dust produced by shotfiring operations in rippings comes from the immediate vicinity of the explosive charge, as the water obviously cannot act as a spray over the complete pile of debris produced. This would also account for the wide variations shown in coal shots, as variations in the nature of the seam may cause it to break into large blocks or crumble up easily, thus producing dust in places remote from the shothole.

Whatever the fundamental reasons, it is clear that the practical reductions in airborne dust counts achieved in ripping shots makes the widespread adoption of water ampoules in these situations very desirable, and where good results are achieved in coal seams, they should also be introduced.

(d) Improvement in the size analysis of the coal product.

As mentioned already on the section on pulsed infusion shotfiring, very closely controlled tests by independent observers are necessary to distinguish between temporary improvements brought about by increased super:vision of a new method, and permanent improvements due to the method itself. The tests of/

of which the results were seen did not satisfy this requirement and no figures are included here for this reason. It is difficult to see what improvement can be effected with a volume of water of 200 - 300 mls. when pulsed infusion shotfiring with the shothole completely filled with water did not make any sensible difference to the size analysis of the product.

Two interesting points came to light which serve to illustrate the point of increased supervision producing temporary improvement. When a colliery in the East Midlands Division introduced pulsed infusion shotfiring in undercut coal, one of the advantages claimed was the elimination of the need for additional shots to bring down coal sticking after the main shots. Two years later, exactly the same advantage was claimed for water ampoules in conjunction with conventional explosives in comparison with pulsed infusion shotfiring.

Again, in tests made to determine any change in size analysis of the product prepared by conventional blasting with and without water ampoules, it is invariably seen from the results that the average weight of charge is reduced when ampoules are used. Obviously, increased production of large coal is a corollary of reduced explosives consumption, but it is difficult to see how the average shotfirer could immediately appreciate the need for reduced charges/

charges. It would appear more than possible that the increased attention which is given to new methods in the trial stages would account for the improvements observed with this method, but only continued experience will show if this is so.

Conclusions

(1) Single shot firing with conventional explosives and methods is still the most widely practised method in this country, accounting for 70% of all shots fired in coal. Simultaneous firing in rounds of up to six shots represents 20% of the remainder and the application of this method is limited in hand loaded faces by the limited speed at which the coal may be removed and supports set.

(2) Alternatives to explosives offer potential safety and economic advantages over conventional explosives, and are likely to become more popular in suitable situations.

(3) The application of pulsed infusion shotfiring is limited, and this method is likely to be used mainly in small scale operations, such as the preparation of stable holes on power-loaded faces.

(4) Off shift firing offers great potential advantages over the conventional method of firing shots on the production shift.

(5)/

(5) The use of water ampoules in conjunction with conventional explosives may, in certain circumstances, considerably reduce the dust produced by shotfiring.

R E F E R E N C E S

(i) Sheathed explosives. Transactions of the Institution of Mining Engineers, CI, 113

(ii) Proceedings of the Shotfiring Convention at Sheffield University, 1957, Part IV - The Ignition Hazard in the Field

(iii) Third Report on Shotfiring in Yorkshire: The Ignition Hazard due to Shotfiring. Transactions of the Institution of Mining Engineers, CXV, 894

(iv) Resistance of Stemming. Transactions of the Institution of Mining Engineers, CV, 62

(v) The Coal Mines (Explosives) Regulations, 1956, 43, (4)

(vi) The Fourth Report of the Committee on Shotfiring and its Alternatives. Transactions of the Institution of Mining Engineers, CXVII, 837

(vii) The Coal Mines (Cardox and Hydrox) Regulations, 1956, 26 (a)

(viii) Dust Prevention and Suppression in Coal Mines. Transactions of the Institution of Mining Engineers, CXII, 81

(ix)/

(ix) Development of a Combined Blasting/Water Infusion Technique for Coal Breaking. Transactions of the Institution of Mining Engineers, CXIV, 87

(x) Mine Explosions: The Current Hazard. Transactions of the Institution of Mining Engineers, CXIV, 862

(xi) Water Infusion of Coal Seams by Explosives. Ann. Min. Belg., LII, 56

(xii) Proceedings of Shotfiring Convention at Sheffield University, 1957, Part V - The Work of the Explosives Branch of the Safety in Mines Research Establishment.

CHAPTER VI

SHOTFIRING IN RIPPINGS

The creation of roads sufficiently large to satisfy ventilation and haulage requirements in longwall work necessitates, in all except the thickest seams, the removal of part of the strata from above or below the coal seam. The importance of this operation is demonstrated by Table XVI, page 83, which shows that in 1958 in N.C.B. mines 14.734 million shots were required in these situations and of these, 96% were initiated singly, simultaneously in rounds of up to six shots or by fuse. The firing of these shots is governed by the same Regulations applying to shots in coal, and the only additional provision imposes an obligation on the shotfirer to use sheathed or equivalent to sheathed explosives in roof rippings within 60 ft. of the coal face in mines in any part of which the use of lights other than permitted lights is unlawful, or in any part of a mine in which safety lamps are being used as a temporary precaution. Alternatives to explosives offer undoubted safety advantages for ripping shots, being free from any danger of igniting methane/ air mixtures, but the application of these devices is limited by the range of materials in which effective blasting and fragmentation may be obtained.

Ripping shots have a much greater potential ignition hazard (i) than do shots in coal, due primarily to the inevitable bed separation in the/

the strata overlying the coal seam creating breaks which may cross the explosive charge and also communicate with accumulations of methane. This problem may be overcome in three ways - by the temporary or permanent elimination of the dangerous conditions or by the development of an explosive which would be completely safe when used in quantities sufficient for any ripping blasting needs. The elimination of the dangerous conditions presents an almost insoluble problem in orthodox longwall working, as the time interval between the removal of the coal and overlying strata inevitably leads to bed separation taking place. However, this may be overcome in three ways, namely, the use of longwall retreating methods where no ripping is normally necessary, the use of headings driven a short distance ahead of the coal face, or the ripping of the floor instead of the roof. Blasting material upwards is wasteful of explosives in comparison with blowing it down. The three methods involve difficulties in application, but they have nevertheless been used, generally for reasons other than the ignition hazard on shotfiring, and are all thus feasible in practice.

The treatment of the dangerous conditions to obtain temporary relief offers an attractive solution to the problem, as it could be applied without an alteration of existing conventional methods/

methods of work. However, there is always the chance that the essential precautions might be neglected just when it was most required, and this must be considered a grave disadvantage of the method. Efforts in this direction have been concentrated by the Safety in Mines Research Establishment into the development and injection of a suitable foam into the shotholes, thus driving any inflammable gases to places remote from the explosive charge. Experimental results (ii) have shown increases in safety in the break gallery test, and limited underground trials have been performed. The use of water ampoules also offers a ready means of including water virtually in contact with the explosive charge, but the probable influence of this in practical conditions is difficult to assess.

The development of an explosive which would be incapable of igniting methane/air mixtures under practical conditions offers the most attractive solution, as it would be foolproof and not interfere with conventional methods of work. Generally, however, increase in safety involves reduction in power and this may impose limitations on the applicability of very low strength explosives in hard rock rippings which are those most likely to give rise to extensive breaks (i).

The development of delay detonators for use in stone mines offered possibilities for increasing/

increasing the efficiency of blasting operations in thick rippings. Where holes are required on more than one horizon, they must generally be fired in separate rounds to achieve economic blasting without the formation of sockets in the upper holes, and this repetition of operations could be eliminated by the use of delay detonators. Shots in rippings are, however, much more likely to give rise to ignitions than shots in stone mines, and at present work is concentrated on attempting to estimate the possible additional hazards introduced specifically by delay blasting, and should these prove to be less serious than at first supposed, the technique may become more widespread.

The dangers arise from the possibility of ignition of methane or coal dust by the detonators and the explosive charge used, and are dealt with in great detail in a paper presented by Grimshaw to the Institution of Mining Engineers - 'The Possible Applications of Short Period Delay Detonators in Safety Lamp Mines' (iii). Briefly, however, they concern the incendivity of the detonators when fired unconfined in methane/air mixtures and the possibility of an early shot in a round removing part of the burden if a later shot, resulting in the latter firing while exposed or partially confined. Few explosives offer high margins of safety in the latter circumstance, and although an ignition could/

could occur only if a body of methane were released by an earlier shot, it is obviously undesirable to introduce any new hazard, however small.

In addition, the method would at present be limited to rippings which could be blasted by up to six shots - the capacity of the largest approved (Type P) exploder - and in these circumstances it is doubtful if the operational advantages gained would outweigh the effort entailed in training shotfirers and arranging for the issue of delay detonators for the limited number of shots fired. Should a larger capacity exploder become available, however, the method may find increasing use in the limited numbers of rippings requiring several rows of holes.

The problem of the reduction in accidents due to projected material from ripping shots should be more amenable to investigation than shots in coal, as the shotfiring operations are generally concentrated both in space and time. However, as indicated in Chapter IV, there is a certain amount of evidence to suggest that these shots present a disproportionately high hazard, suggesting that the location and direction of the shotholes may be of overriding importance, when considering accidents from this cause.

Conclusions

(1)/

(1) Over 96% of all shots in rippings are fired with conventional explosives initiated by instantaneous detonators fired singly, simultaneously in rounds of up to six shots or by fuse.

(2) Alternatives to explosives are employed only to a limited extent in rippings, probably because of the difficulty in obtaining effective fragmentation of hard strata with these devices.

(3) Delay firing is used to a small extent, and the extension of this technique in rippings will be influenced by the development of a suitable exploder of 12 - 15 shots capacity, the number of rippings which require shots on several horizons, and the inconvenience of the additional training of shotfirers and issue of delay detonators for the comparatively small number of shots fired.

(4) The complete elimination of ignitions resulting from shotfiring in rippings can be achieved only on the development of a completely safe explosive, but the use of methods of work which do not allow bed separation to take place before firing could provide a means of reducing the hazard in especially dangerous situations.

R E F E R E N C E S/

R E F E R E N C E S

(i) Proceedings of the Shotfiring Convention at Sheffield University, 1957, Part IV - The Ignition Hazard in the Field

(ii) Proceedings of the Shotfiring Convention at Sheffield University, 1957, Part V - The Work of the Explosives Branch of the Safety in Mines Research Establishment

(iii) Grimshaw, H. C. 'Possible Applications of Short Delay Detonators in Safety Lamp Mines'. Transactions of the Institution of Mining Engineers, CXIII, 31

CHAPTER VII

SHOTFIRING IN STONE MINES

The recent introduction of horizon mining into this country has necessitated the drivage of much longer roads through solid rock strata than was necessary with the traditional 'in the seam' methods formerly employed, and in 1958 in N.C.B. mines 5.614 million shots or 6% of the annual total were fired in these situations. As the shotfiring operations are very localised in comparison with coal and ripping blasting, they form a suitable subject for specialist study and have in fact been highly developed to increase the overall speed of stone mine drivages.

The problems encountered in the blasting operations are almost purely technical and thus outwith the scope of this work but certain general principles may be stated. The rock strata is not cut or sheared, but the initial shots are located and loaded to provide an additional free face to which other shots may blow. Shots are then fired in groups successively further from the free face until the required tunnel shape has been formed. The development of half-second and millisecond delay detonators has enabled all the shots to be charged, stemmed and fired at once, effecting considerable savings in the time taken by shotfiring operations and speeding up the overall drivage rate obtained. When large numbers of shots/

shots are initiated simultaneously a Type N exploder, unapproved for general use underground, must be used and the conditions attached to the exemptions given must be rigidly observed if safety is to be ensured. In 1957, two explosions in stone mines (i) were attributed to incendive sparking at defective cable joints when a high capacity dynamo condenser exploder was in use, coupled with the existence of considerable undetected concentrations of methane existing at the roof of the tunnels.

The incidence of shotfiring accidents in stone mines is not high (Table XVII) due probably to the unusually favourable circumstances in these situations. The concentration of shotfiring with the extensive use of delay blasting results in a reduction in exposure to risk and a correspondingly reduced temptation to neglect the precaution of taking cover, and the single means of access to the shots eliminates any dependence on sentries, as the shotfirer will normally perform this duty himself.

R E F E R E N C E S

(i) Explosion at Golborne Colliery, Lanca-:shire - Report by G. Hoyle, C.M.G., H. M. Divisional Inspector of Mines and Quarries

Explosion at Risehow Colliery, Cumberland - Report by H. Hyde, H. M. Divisional Inspector of Mines and Quarries.

CHAPTER VIII

THE CAUSES OF SHOTFIRING ACCIDENTS

In Chapter I, it was shown that all shotfiring accidents could be divided into two classes, namely, those which could be attributed to some failure of the explosive or ancillary equipment to attain technical perfection, and those due to failure of the human element. The elimination of the former class is a purely technical problem and the current indication, shown on Graphs I and II, is that a considerable measure of success has been achieved in this field. The latter class of accident, which is shown by Graph II to form a very large and increasing percentage of the total, may be further subdivided into those in which a breach of the Regulations was the main contributory cause and those in which the shotfirer, through a faulty estimation of all the factors involved, failed to ensure that everything was safe for the firing of the shot or shots. However, this subdivision cannot be carried out with any reasonable certainty or accuracy as accidents may result from a combination of factors and the influence of each will be impossible to determine. It is convenient, however, to treat such accidents as being due to one or other of the two causes, and the subdivision including accidents due to breaches of the Regulations are considered first.

At the present time any person firing shots in collieries where permitted explosives only may be used must have

(a)/

(a) a gas testing certificate obtained within the previous five years,

(b) a shotfirer's certificate, granted by the Ministry of Power on the recommendation of the Mining Qualifications Board and spent not less than five shifts in practising shotfiring in a mine under the close personal supervision of a shotfirer, or

a First or Second Class Certificate of Competency, or

held a service certificate under Paragraph 6 of Article 12 of the 1951 Explosives in Coal Mines Order, stating that he regularly fired shots in a mine to which Part II of the 1934 Explosives in Coal Mines Order applied,

(c) attained the age of twenty-two years, or twenty-one years if he holds a General Certificate in Mining or higher qualification, and,

(d) had three years underground experience with not less than eighteen months at the coal face, with not more than six months in stone mine drivages.

Although anyone may sit the Shotfirers' Examination, it is now usual for whole time residential or part-time evening classes to be organised and these deal in great detail with the requirements of the Regulations. The final examination ensures that all who pass have a sound working knowledge of their duties and obligations and it is impossible that any shotfirer breaking the law does so unwittingly. Nevertheless/

Nevertheless accidents caused by breaches of the Regulations continue to occur, and it is clear that some reason unconnected with faulty training must exist.

Throughout the period of study, ample opportunity has been afforded by visits to collieries in different Areas and Divisions to observe the standard of practice achieved in shotfiring. Even when there was no rush of work, in no case seen did a shotfirer carry out the provisions of the Regulations in full, and frequently the faults of omission or commission were such as to be conducive to accidents. The most common breaches observed were laxity in gas testing procedure, the firing of more than one shot with a single shot exploder, the use of very small quantities of stemming, the firing of shots wholly or partially in solid coal and the charging and stemming of several shots when single shot firing was in use. Frequently any attempt at break detection was purely token. One very marked feature was the stress laid on different operations at different collieries. Thus at one pit the shotfirer did not take his flame safety lamp on to the face but would not fire any shot unless the undercut was properly gummed, while at another this was reversed, with gas testing being carried out before shots were fired in the solid coal.

However/

However, undoubtedly the most disturbing feature observed was that the malpractices went on with the silent approval, and sometimes active encouragement, of the section oversman and deputy. While it is obvious that only a sense of responsibility in the shotfirer himself can ensure that all the necessary operations are carried out for every shot, it is equally clear that some supervision must be imposed. No uniform policy exists throughout the National Coal Board but Explosives Engineers, concerned mainly with the economic aspects of shotfiring, have been appointed in certain Divisions and at Area and pit levels Large-coal Officers and Safety Officers include the supervision of shotfiring practice in their duties. It is obvious that any control exercised by those officials can have only a limited effect, as any section visited will probably be forewarned.

The committee on the precautions necessary to ensure safety in the use of explosives in coal mines (i) suggested that the employment of a specialist oversman, who would concern himself solely with the use and handling of explosives, would yield worthwhile safety advantages by imposing closer control on shotfiring practice. It has been repeatedly shown by the technical representatives of explosives manufacturers (iv) that increased supervision of shotfiring operations results in considerable reductions in explosives consumption/

consumption and increases in the percentage of round coal obtained. Unfortunately, these improvements are lost when the control is removed. It must be stressed that the economic and safety requirements of shotfiring are very similar, and that although débris may be projected from any shot the possibility is increased should the hole be overcharged, understemmed or badly placed. These factors also lead to degradation of the product and high explosives consumption. Undoubtedly the appointment of an 'explosives oversman' at colliery level to supervise shot-:firing would help to eliminate these (and other) malpractices, and bring about an attendant reduction in the accident hazard.

In circumstances where the method of work adopted by the management results in the work of shotfiring being unevenly distributed throughout the shift, the problem becomes very much more complicated and it is then not possible to place the blame for failure to observe the provisions of the Regulations solely on the shotfirer and other section officials. The requirements relating to the firing of single shots are very comprehensive and it is obvious that the fulfil-:ment of them all must take a considerable time. No previous record of a time test could be found and it was decided that to obtain a measure of the difference (if any) existing between rates of shotfiring achieved in practice and allowed in theory from consideration/

consideration of the results of the time test, a series of observations would be taken at an orthodox hand filled longwall face.

The conditions were felt to offer a reasonable compromise, without being excessively easy or difficult. The working height of the seam was four feet six inches and the full dip of one in five was along the face line towards the main gate. The shotfirer, who held a Shotfirer's Certificate (ii) had over ten years experience at his job and was fully familiar with the requirements of the Regulations. Unfortunately, due to the pressure of production only two time tests could be carried out, with the results given in Table XX.

This test imposes a maximum, under these specific conditions, of approximately five shots per hour, if some allowance is made for assistance received from face workers in stemming the hole, removing tools, etc.

The second part of the work consisted of taking a note of the time at which every shot was fired, for four successive production shifts. From the average time intervals between the shots the corresponding rate of shots per hour was calculated (Tables XXI - XXIV) and plotted against the time throughout the shift (Graphs XXVIII - XXI). The plots obtained show, as might be expected, a marked similarity and two peaks are evident before and/

TABLE XX

Results of Two Time Tests of the Firing of a Single Shot (not in a group) Electrically

Operation	First Test Mins.	First Test Secs.	Second Test Mins.	Second Test Secs.	Mean Mins.	Mean Secs.
1. Precharging examination	2	22	2	45	2	34
2. Cleaning shothole, testing for breaks	0	23	0	21	0	22
3. Comparison of the depth of shothole with undercut	0	8	0	9	0	9
4. Sealing the back of hole with a plug of stemming	0	8	0	8	0	8
5. Inserting main charge into the hole	0	27	0	21	0	24
6. Preparing the primer charge and inserting into the hole	0	25	0	25	0	25
7. Stemming the shothole	1~~9~~	21	1	26	1	23
8. Posting and instructing sentries, withdrawing men and tools to the firing station	1	30	1	42	1	36
9. Prefiring examination	1	26	1	34	1	30
10. Firing the shot:						
(a) Connecting detonator leads to cable	0	8	0	9	0	9
(b) Running out cable to firing station	0	50	0	46	0	48
(c) Connecting cable to exploder, checking that everyone is in cover, shouting 'Fire', timing the key and detaching leads	1	14	1	8	1	11
11. Making the after firing examination	0	45	0	50	0	47
12. Recalling workmen and issuing instructions	0	45	0	53	0	49
13. Coiling the cable	0	27	0	28	0	27
Total time taken	12	19	13	5	12	42

TABLES XXI - XXIV

The times of firing shots throughout four successive shifts on a longwall face, the interval between shots, the average intervals between shots and the corresponding rate of firing in shots per hour.

TABLE XXI - FIRST DAY

Time of Firing a.m.	Interval Between Shots (Min.)	Average Time Between Shots (Min.)	No. of Shots per Hour
7.40			
	30		
8.10		20	3
	10		
8.20		10	6
	10		
8.30		$17\frac{1}{2}$	3.4
	25		
8.55		$17\frac{1}{2}$	3.4
	10		
9.05		10	6
	10		
9.15		$7\frac{1}{2}$	8
	5		
9.20			
	Piece-time		
9.50			
	30		
10.20		20	3
	10		
10.30		$7\frac{1}{2}$	8
	5		
10.35		$6\frac{1}{2}$	9.3
	8		
10.43		$6\frac{1}{2}$	9.3
	5		
10.48		7	8.5
	9		
10.57		$8\frac{1}{2}$	7
	8		
11.05		$16\frac{1}{2}$	3.6
	25		
11.30		16	3.6
	7		
11.37		7	8.5
	7		
11.44		$21\frac{1}{2}$	2.8
	36		
12.20		28	2.2
	20		
12.40			

TABLE XXII - SECOND DAY

Time of Firing a.m.	Interval Between Shots (Min.)	Average Time Between Shots (Min.)	No. of Shots per Hour
9.10		30	
	30		2
9.40			
	Piece-time		
10.25			
	15		
10.40		10	6
	5		
10.45		$7\frac{1}{2}$	8
	10		
10.55		$8\frac{1}{2}$	7
	7		
11.02		6	10
	5		
11.07		$16\frac{1}{2}$	3.6
	28		
11.35		$16\frac{1}{2}$	3.6
	5		
11.40		5	12
	5		
11.45		5	12
	5		
11.50		6	10
	7		
11.57		$7\frac{1}{2}$	8
	8		
12.05		14	4.25
	20		
12.25			

TABLE XXIII - THIRD DAY

Time of Firing a.m.	Interval Between Shots (Min.)	Average Time Between Shots (Min.)	No. of Shots per Hour
8.00			
	10		
8.10		$7\frac{1}{2}$	8
	5		
8.15		$7\frac{1}{2}$	8
	10		
8.25		10	6
	10		
8.35		10	6
	10		
8.45		15	4
	20		
9.05		$12\frac{1}{2}$	4.8
	5		
9.10		10	6
	15		
9.25		10	6
	5		
9.30			
	Piece-time		
10.10			
	10		
10.20		10	6
	10		
10.30		$22\frac{1}{2}$	2.7
	35		
11.05		$22\frac{1}{2}$	2.7
	10		
11.15		$7\frac{1}{2}$	8
	5		
11.20		5	12
	5		
11.25		5	12
	5		
11.30		$7\frac{1}{2}$	8
	10		
11.40			

TABLE XXIV - FOURTH DAY

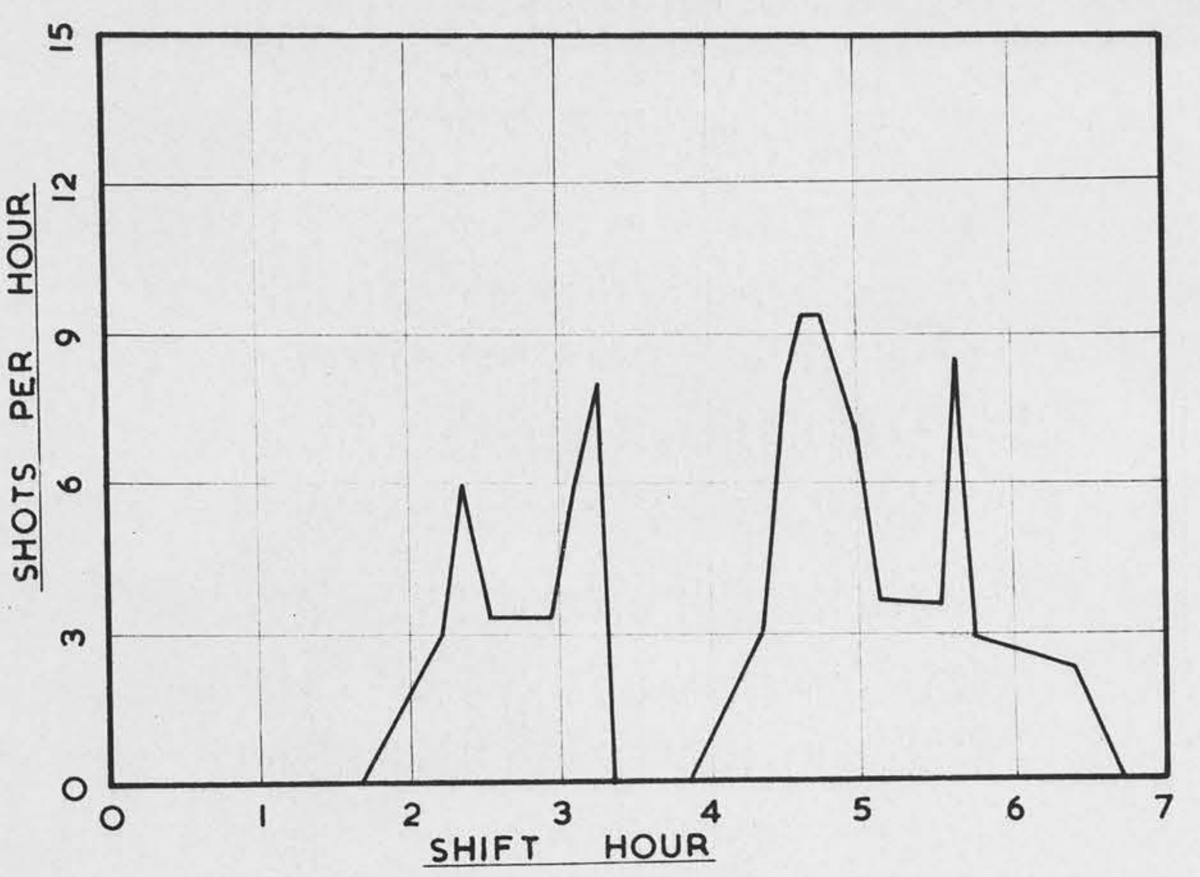
Time of Firing a.m.	Interval Between Shots (Min.)	Average Time Between Shots (Min.)	No. of Shots per Hour
7.38			
	9		
7.47		19	3.2
	29		
8.16		21½	2.8
	14		
8.30		10½	5.7
	7		
8.37		13½	4.4
	20		
8.57		13	4.6
	6		
9.03		8½	7.0
	11		
9.14		10½	5.7
	10		
9.24			
	Piece-time		
10.32			
	5		
10.37		5	12
	5		
10.42		5½	10.9
	6		
10.48		6½	9.3
	7		
10.55		13	4.6
	19		
11.14		18½	3.2
	18		
11.32		11½	5.2
	5		
11.37		6	10
	7		
11.44		7½	8
	8		
11.52		15½	3.9
	23		
12.15			

GRAPHS XXVIII - XXXI

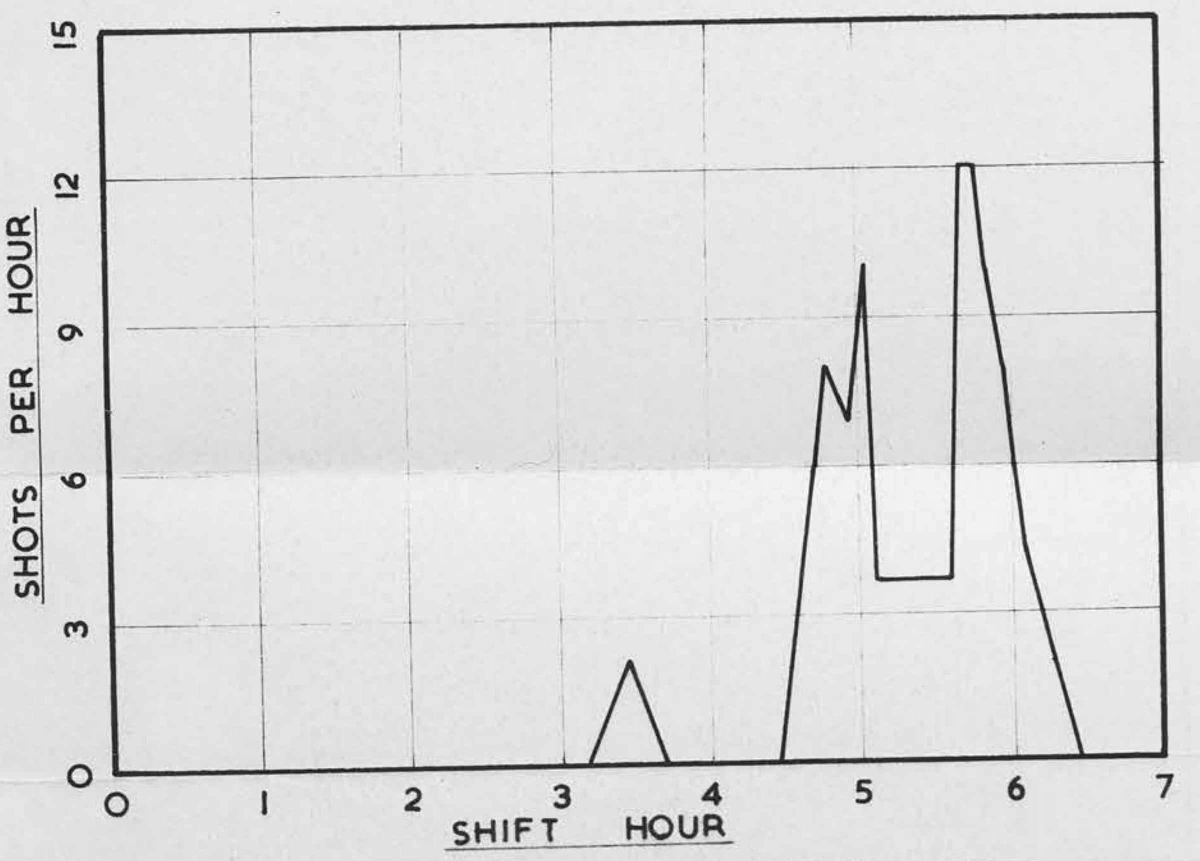
THE SHOTFIRING RATE ON AN ORTHODOX HAND-FILLED LONGWALL FACE PLOTTED AGAINST THE SHIFT HOUR FOR FOUR SUCCESSIVE PRODUCTION SHIFTS

SHOTFIRING RATE

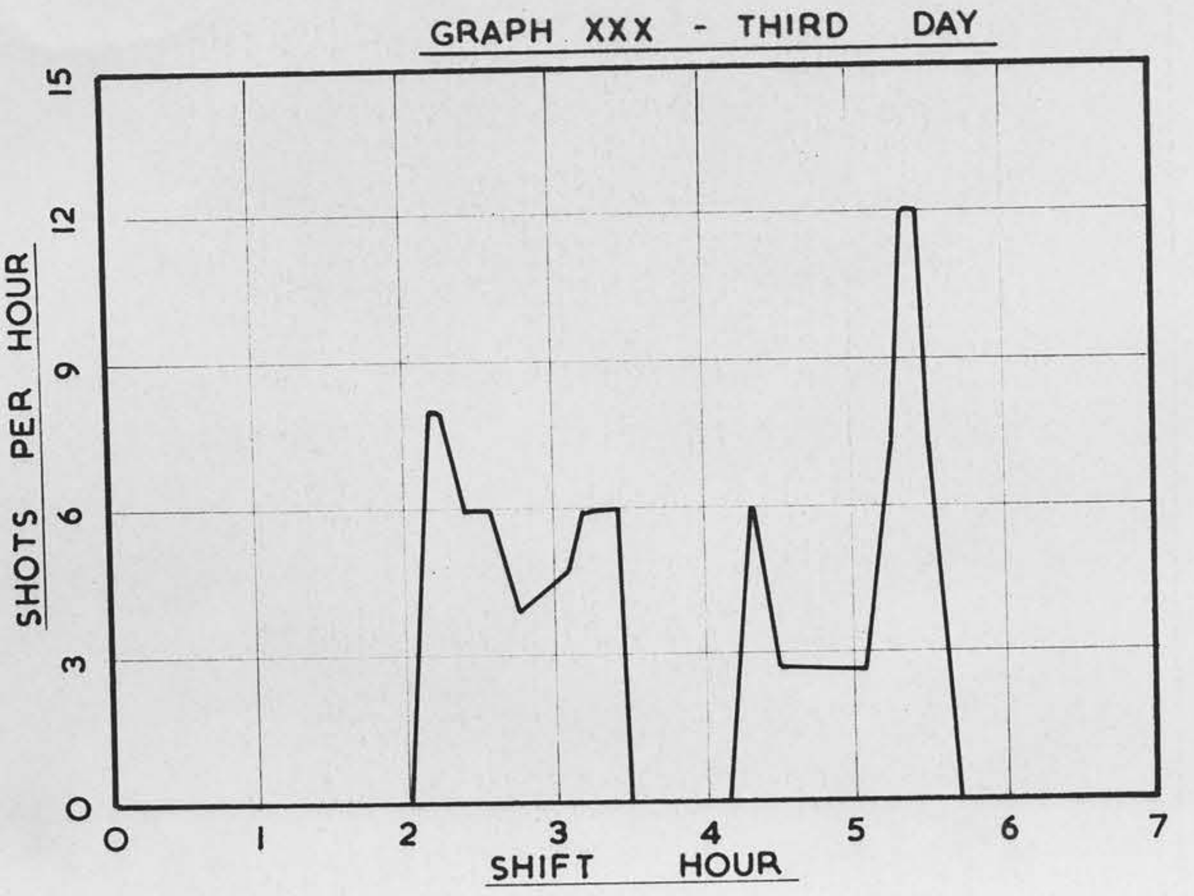
GRAPH XXVIII - FIRST DAY



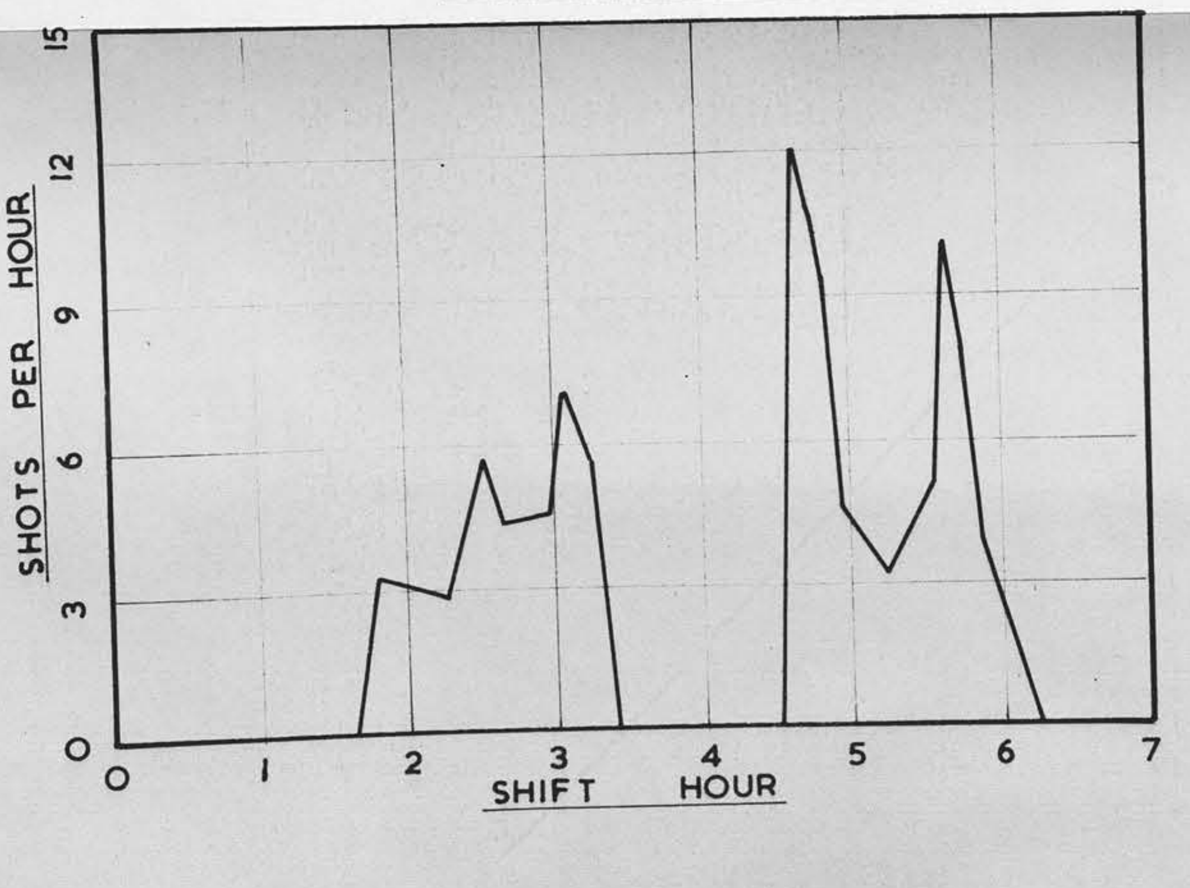
GRAPH XXIX - SECOND DAY



GRAPH XXX - THIRD DAY



GRAPH XXXI - FOURTH DAY



and after piece time, except on the second day when a plant breakdown interfered with the normal working of the section. The Graphs reflect the method of work in the section, whereby the shotfirer worked his way down the group of men he served firing breaking-in shots in each place. After an interval of comparative quiet while the coal prepared by these shots was filled out, the journey was repeated and another shot or shots was fired for each man. The pattern is again revealed after piece time with two high peak values being shown.

Obviously, it is impossible that the work of shotfiring can ever be evenly distributed through-:out the shift, but the disparity between the observed and calculated firing rates is extremely disturbing. Reference to the Tables and Graphs shows that several shots were fired after a time interval of five minutes from the preceding shot, allowing only the essentials of charging and stemming of the shot to be carried out without any precautionary examinations. In addition, it should be noted that a temporary change in the nature of the coal in the section had resulted in only 20 - 25 shots being fired by each shotfirer, and a return to normal conditions would probably result in the peaks being maintained over longer periods.

Although/

Although these tests could not be repeated on other faces the same feature of alternately high and low shotfiring rates was observed, and it is difficult to see how this could be eliminated when normal shotfiring practice is adopted on orthodox hand filled longwall faces. Obviously, men working at approximately the same speed on similar tasks must be ready for shots at approximately the same time. While similar tests would have to be carried out on every face on which an accident occurred to establish a relationship between 'rush periods' and the occurrence of accidents, it is inevitable that the circumstances requiring such haste must predispose the face personnel to neglect the essential precaution of taking cover. This must be considered an inherent disadvantage of orthodox longwall working and as long as human nature remains unchanged accidents resulting from this cause will never be eliminated. The use of alternatives to explosives, permitting high shotfiring rates to be legally maintained, (Chapter V), offers a palliative to the problem, and the introduction of off-shift firing, where applicable, would provide a complete solution.

Although the great majority of accidents are caused by projected material striking people in shelter shown by the occurrence of the accident to have been inadequate, a small number are caused each year by a misunderstanding between the/

the shotfirer and the sentry, or the sentry failing to stop someone walking towards the shot, in each case resulting in someone being at or near the shot when it goes off. In certain collieries the shotfirer is provided with a red plastic disc which he fits over the sentry's cap lamp and this serves to remind him constantly of his duties. Even should this fail to have the desired effect it is unlikely that anyone would walk past without seeing the disc and enquiring into the reason for it. There seems to be no good reason why this practice could not be extended to all collieries with a consequent elimination, or at least reduction, in one of the hazards of shotfiring.

Although breaches of the Regulations take place when virtually every shot if fired, these may not be the main or even a contributing cause when an accident occurs, and the Chief Inspector of Mines and Quarries in his annual report for 1958 stated that 'most of the (shotfiring) accidents which occurred during the year were caused by blows from projected material in circumstances where there had been an error of judgment in determining the danger zone of the shot or where persons had not withdrawn to proper shelter'. While the occurrence of an accident obviously shows that an error of judgment has occurred, it is felt that the statement above rather evades the basic problem - the difficulty in many/

many situations of finding any suitable cover.

The actual wording of the Regulations relating to the taking of shelter is:

(1) Any shotfirer proposing to fire a shot shall before firing determine the danger zone likely to be created.

(2) No shotfirer shall fire any shot unless he has:

(a) at each entry to the danger zone posted a sentry or placed an appropriate fence conspicuously marked with the words 'danger' and 'shotfiring';

(b) ensured that all persons have withdrawn from that zone or have taken proper shelter; and

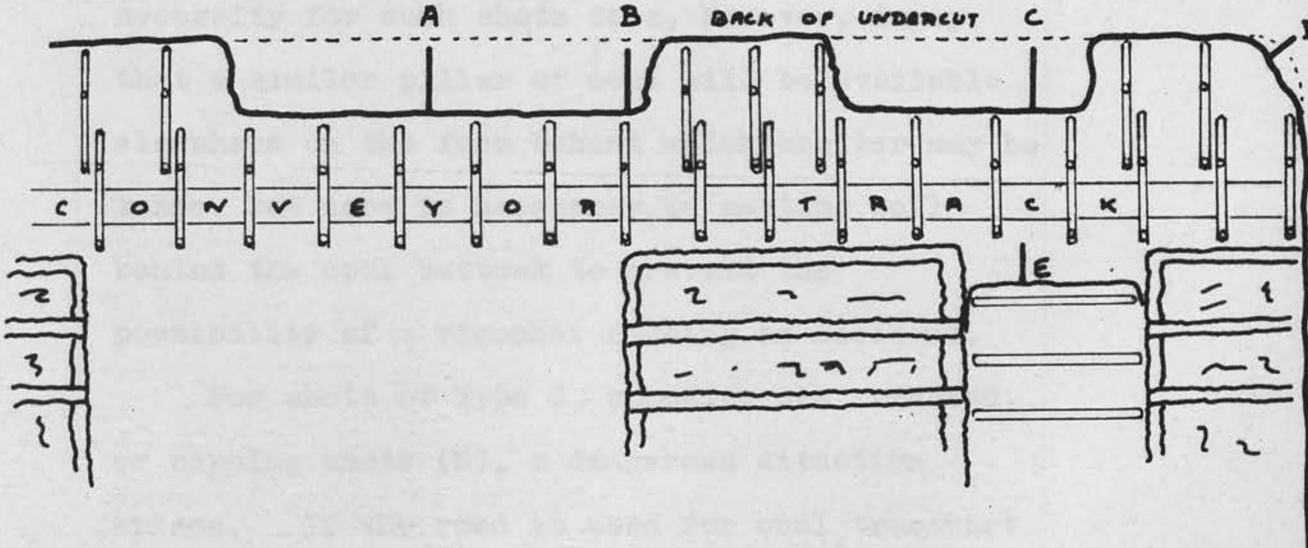
(c) himself taken proper shelter.

Obviously, when explosives are in use, any energy in excess of that required to dislodge and partially break up the material being blasted must be dissipated at least in part in the degradation and projection of coal or stone. In exceptional cases the energy may be concen-:trated on a small mass of stemming and coal or stone and this may be blown with great force over considerable distances. Reference to Figure III illustrates the difficulty that a shotfirer must face in attempting to ensure that everyone is in shelter or outwith the danger zone when shots are fired on a longwall face. For breaking-in shots of Type A the only possible way in which solid cover may be interposed between the shot and the face personnel/

FIGURE III

PLAN OF PART OF A LONGWALL FACE, SHOWING THE LOCATION AND DIRECTION OF THE SHOTHOLES FOR

A - BREAKING-IN SHOT
B - INTERMEDIATE SHOT
C - SHOT OPPOSITE THE ROADHEAD
D - SHOT IN THE CORNER OF THE FACE
E - SHOT IN THE RIPPING



personnel is for the latter to retreat down the roads to the face, and this rapidly becomes impracticable as the shotfirer works towards the middle of the face. It is of course true that, because of the constraining action of the coal on either side of the shot, any debris projected will tend to blow into the pack or waste opposite, but the remote possibility of a ricochet being deflected at an acute angle up or down the face must always remain.

Shots of Type B are required after the coal prepared by the breaking-in shot has been cleared. As indicated in Chapter V, the hole placement may not be ideal and the burden may be very unbalanced - obviously circumstances which could lead to the projection of material. The necessity for such shots does, however, imply that a similar pillar of coal will be available elsewhere on the face behind which shelter may be taken, but care is necessary in getting well behind the coal buttock to prevent the possibility of a ricochet causing an accident.

For shots of Type C, opposite the roadhead, or ripping shots (E), a dangerous situation arises. If the road is used for coal transport several men will be working in it and all will be exposed to the risk of an accident. While it is widely appreciated that distance alone gives protection from these shots only when the separation between man and shots is carried to impracticable/

impracticable extremes, there is an unfortunate tendency for men to be satisfied with shelter of very doubtful efficiency, and the very large numbers of shots fired will inevitably result in the gaps in poor quality protection being found by projected material.

In the corners of the face, shots of Type D, with the holes bored at an angle as shown, must generally be fired and these, being at least in part over solid coal with a second free face provided by the removal of the neighbouring coal, are naturally heavily charged. Because of the shothole placing and direction, any material projected may ricochet either out the road or up the face, and as most of the coal will normally have been cleared when these shots are fired, there will be little shelter available on the face.

Thus the problem of ensuring that everyone is in proper shelter or outwith the danger zone when shots are fired in coal on longwall faces is by no means easy, as there is little shelter on the face and that available in the roads may not be suitable. In fact, considering the amount of cover which can be utilised, the accident rate of approximately one per 300,000 shots can only be considered as remarkably low.

The use of shelters offers a possible means of reducing this hazard, by introducing additional protection capable of deflecting or stopping/

stopping projected material. All such devices fall into two main categories - those which provide limited protection for a complete area by occupying the entire cross-sectional area between the shot and the personnel, and those which give complete protection for a much more limited area, by means of a solid shield close enough to the personnel to prevent the possibility of any ricochet being deflected to hit them. The first type are, by their nature, restricted in use to circumstances where comparatively settled conditions exist, and to prevent undue restriction of the ventilation, they must be made of material with a large number of openings, albeit of small dimensions. For these reasons they are used only in roads leading to the face, and are constructed as curtains so that the lower portions may be drawn to the side when not required (iii). Obviously these devices cannot give complete protection and can be considered only as an added safeguard to more solid types of shelters. Shotfiring shelters of the second type may be made of any material available at the colliery or in the section, and old conveyor belting or wooden boards, with or without metal reinforcement, are frequently used. The design normally allows the structure to be folded away when not in use.

When/

When considering the undoubted safeguards which the use of shotfiring shelters will bring when properly constructed and used, it is essential to bear in mind certain inherent possible disadvantages. One of the fundamental maxims in industrial accident prevention is that a safety device, to be effective, must be outwith the control of the workmen, and it is obvious that no shelter can ever satisfy this requirement. In the East Midlands Division, shotfiring curtains are provided in virtually every gate road in longwall workings, but in some faces visited they were not properly erected and the lower sections of netting were lying at the side of the gate, obviously out of use, and this would certainly seem to bear out the truth of the above contention. Nevertheless, if no suitable shelter exists the shotfirer is obviously placed in a most unsatisfactory position and he will be forced to fire shots when the legal requirements are not, and cannot be, satisfied. However, in roads used for coal transport manholes must be made and these, if properly constructed, provide complete protection. In such circumstances the provision of shelters merely duplicates cover already existing, and it is difficult to see why a shotfirer should go to the trouble of erecting a shelter if he will not go into a manhole/

manhole. On roads where manholes are not required by law, it is unlikely that even the most rudimentary cover will exist and in such circumstances a much stronger case for the provision of shelters can be put forward. Nevertheless, the same purpose could be served more efficiently by the provision of an occasional manhole, as few men will normally be working in such roads (either tail gates or coal headings) when shotfiring takes place. In stone mines shotfiring is now normally carried out only once per shift and the temptation to skimp the precaution of taking cover is correspondingly reduced. When high capacity exploders are in use, the exemption normally specifies a minimum distance from the face to the firing station in itself adequate to ensure safety, but again in long stone drivages manholes will be provided, giving adequate shelter without the need for any additional protection.

Again, shelters to be effective have to be strong enough to withstand the most severe blow likely to be encountered. Shotfiring accidents are exceptional occurrences and it is essential to take into account every possible contingency when designing shelters. Thus, should a heavily charged shot blow out, any stones in the stemming material will be capable of penetrating shelters adequate for normal conditions/

conditions. It has also been suggested that the existence of shelters might lead to abuses, as a false sense of security created might lead to their use too close to the shot being fired. Those two factors are not distinct as the distance from the shot and the strength of the shelter must be to a certain extent interdependent. This point is extremely well illustrated by an example given in the Annual Report of the Chief Inspector of Mines and Quarries for 1958, which cites the case of a shotfirer who was killed when he fired a ripping shot from behind a sheet of corrugated iron thirty-two yards from the shothole. The material penetrated the sheet with sufficient reserve of energy to puncture his heart.

The combination of the disadvantages of the difficulty of making a shelter suitably strong, the uncertainty of its use if provided, and the possibility of abuse makes it difficult to imagine that shotfiring shelters will ever be used to any extent on the coal face. The two latter disadvantages in particular apply even more strongly in this situation than in the more pleasant environmental conditions of gate roads.

<u>Conclusions</u>

(1) Shotfiring accidents resulting from failure of the human element may in theory be divided into those in which a breach of the Regulations was the main contributory cause, and those due primarily/

primarily to errors of judgment. In practice, it is generally impossible to estimate the significance of the individual factors, as both causes may contribute to individual accidents.

(2) The general standard of shotfiring practice is low, and this is due both to apathy on the part of the section officials allowing the high standard demonstrated and achieved in the training period to deteriorate, and the present system of orthodox hand filled longwall working which prevents the shotfirer from working at an even rate throughout the shift.

(3) In orthodox longwall working it is frequently impossible for the shotfirer to ensure that everyone is in proper cover prior to the firing of the shot. This factor, combined with the impossibility of providing suitable artificial shelters in the cramped conditions on the coal face, would appear to indicate that shotfiring accidents will never be eliminated when conventional shotfiring practice is adopted on orthodox hand filled longwall faces.

(4) The use of shelters offers considerable theoretical advantages but the possibility of their non-use or abuse, coupled with the difficulty in estimating the strength which is required militates against their use in practice. The only absolutely safe cover is solid rock between the shot and the personnel, sufficiently close to the latter to prevent rico-

chets/

:chets causing accidents, and this is provided efficiently by manholes. The provision of these even when not legally required would seem more desirable than the use of shelters of perhaps doubtful efficiency.

REFERENCES

(i) Report of the committee of enquiry into the precautions necessary to secure safety in the use of explosives in coal mines, 1950.

(ii) Coal Mines (Explosives) Regulations, 1956. 5 (b).

(iii) Safety curtains for use during shotfiring. N.C.B. Information Bulletin No. 54/109.

(iv) McCormick and Partington. Recent experiences in blasting methods to improve production of round coal. Journal of the Wigan and District Mining and Technical College, Vol. 1, 1959.

CHAPTER IX

EXPLANATION OF LOCAL ACCIDENT RATE VARIATIONS

Chapter IV, dealing with the statistical study of the distribution of shotfiring accidents, showed that most National Coal Board Divisions could be considered average when compared with the rest of the country. In other words, most of the variations in accident rates which occurred could be explained by the fluctuations which might be expected in any random distribution of numbers. However, two Divisions - the Scottish and the North Eastern - showed marked divergences from average, Scotland being very bad and the North East very good when compared with the rest of the country. A further analysis of the Scottish Division showed that no Area accounted for an excessive proportion of the total, but that the Central East Area had an indicatively lower accident rate than the rest of the Division. Any analysis of shotfiring must include an attempt at the explanation of these divergences from average conditions.

The general comments of Chapter VIII apply, with certain reservations, to all geographic areas, and the explanation of indicative or significant variations in accident rates must lie in differences in working conditions, methods of shotfiring and standards of care particular to the area considered. Any attempt at analysis is difficult as these factors may operate/

operate individually or collectively, and the same overall position may be achieved in a variety of ways. Thus an average accident rate may result from the combination of an inherently dangerous practice and high standards of care, or a safe practice and low standards of workman-:ship. This precludes what would otherwise have been a simple comparison of the good and bad Divisions to determine which controlling factors influenced the occurrence of accidents.

Off-shift firing, and the use of portable shelters were shown in Chapter VIII to offer possible means of reducing shotfiring accidents and, with the co-operation of the National Coal Board, a circular was prepared and sent to each Division, enquiring into the extent of the use of these techniques. The replies are given in Table XXV, exactly as received, and the comments in the column headed 'Accident Position' are derived from consideration of Graphs VIII - XVI, pages 66 - 70. The results are in a sense disappointing as no absolutely clear cut conclusions can be drawn. However, it is at once evident that the explanation of the high accident rate in Scotland does not lie in the very limited use of off-shift firing or non-use of portable shelters, as the South West, combined West Midlands/South East and North West Divisions with average or better than average accident rates share these features. Comparison of

Graph/

TABLE XXV

The use of Off-shift Firing and Portable Shelters, and the Accident position of the Divisions of the National Coal Board

Division	'Off-shift' Shotfiring	Use of Portable Shelters	Accident Position
Scottish	Practiced in one Area out of the eight in Division.	Not used.	Very bad
Durham	Quite widely practiced, particularly in East Coast pits where the shotfirers and sometimes a few fillers (to clean up the gummings) go on ahead of the main coaling shift and get sometimes as much as three-quarters of the shotfiring done.	Portable shelters have been extensively used in one Area for some time and are now being introduced on a larger scale into all Areas both for gate roads and for drifts.	Average
Northern (N & C)	Is practised in certain Areas, but the practice varies considerably from one pit to another and reliable figures of the proportion of faces where 'off-shift' firing is practiced are difficult to obtain.	The use of portable shelters is increasing almost daily and the position at the end of Sept., 1959 was that three Areas showed 140, 72 and 216 portable shelters in use where 13, 7 and 20 collieries respectively were involved.	Average
North Eastern	Generally speaking, the majority of the face shotfiring takes place two hours or so before the filling shift.	Only a few pits are using portable shelters in gate roads and none at all in stone drifts. The others tend to make full use of a specially constructed refuge hole.	Very good
North Western	Very limited application.	Not used	Better than average
East Midlands	'Off-shift' shotfiring is in respect of approximately 10% of total operations.	Portable shelters are used universally in gate roads and in stone drifts where limited firing is carried out, i.e. up to 6 shots per round. For heavy firing in large drifts, i.e. rounds up to 50 shots, it is customary for the firing point to be in a roadway different from the one in which the shots are fired.	Average
West Midlands & South Eastern	Practiced at approximately 20% of the collieries, but this figure is being reduced.	Very limited	Average
South Western	Only to a limited extent.	Only to a limited extent	Average

Graph XIV and the comments in the table on the East Midlands Division indicates that the provision of portable shelters does not appear to effect any reduction in shotfiring accidents. It should be noticed that the word 'used' in the table should be accepted with the reservations indicated in Chapter VIII. Only one Division, the North Eastern, uses off-shift firing for the majority of coal preparation shots and this coincides with a significantly low accident rate, indicated on Graph XV. It is difficult to assess from the reply received from Durham the exact extent of the use of off-shift firing in that Division.

In the Scottish Division off-shift firing is confined to the Central East Area and this, for the rather limited period which could be examined, coincides with indicatively low accident figures, shown on Graph XVIII. It is interesting to note in Table XXVI that in three out of the four years examined, the accident rate from falls of roof and side at the face, based on the tonnage of coal produced was lower in the Central East Area than the rest of Scotland and that the use of the method does not apparently result in dangerous conditions being created at the coal face. It must, however, be emphasised that the shotfiring accident rate in the Central East Area is indicatively low only when compared with the rest of the Division.

If/

TABLE XXVI

A Comparison of the Accident Rates due to Falls of Roof and Side at the Face, for the Years 1954-57, for (a) Central East Area of the Scottish Division and (b) the rest of the Scottish Division

	Central East Area			Rest of Scottish Division		
	Accidents from falls of roof and side at the face	Output in tons x 10^6	Accident rate per million tons	Accidents from falls of roof and side at the face	Output in tons x 10^6	Accident rate per million tons
1954	425	3.39	125.5	2460	18.84	130.5
1955	378	3.29	115.0	2333	18.42	126.5
1956	347	3.10	112.0	2275	18.04	126.0
1957	333	3.10	107.5	1870	17.89	104.5

If expectations are calculated from the accident rate for the rest of the country, as in Table XXVII, and plotted against the observation with the usual confidence units superimposed, as in Graph XXXII, it is seen that only one year falls outside the region of no significance. This provides an example of the simultaneous operation of two conflicting factors - the Area rate being reduced below the Divisional average by off-shift firing, and increased to the National average by the effect of the overall Scottish Divisional influence, considered in greater detail below.

As already mentioned, no explanation for the high accident rate in Scotland can be found from consideration of the use of off-shift firing and portable shelters, and the solution must lie in purely local differences in conditions, methods of work or standards of practice. In Scotland, generally speaking, mining conditions are rather worse than the rest of the country and thin seams, hard coals, faulted ground and high strata inclinations are frequently encountered. Thin seams require more, although lighter, shots per ton of output than do thick seams, and the combination of the frequent need to take shelter and the difficulty encountered in moving in seams less than two feet six inches thick will inevitably lead to slackness in taking shelter. This also applies/

TABLE XXVII

For each year of the period 1954-57, the observed number of accidents (O) and the number of shots fired (N) in the Central East Area of the Scottish Division are shown. The National totals of shots (T) and accidents (A) are also given. Expectations (E) are then calculated for the Central East Area, based on the average accident rate for the rest of the country, i.e.

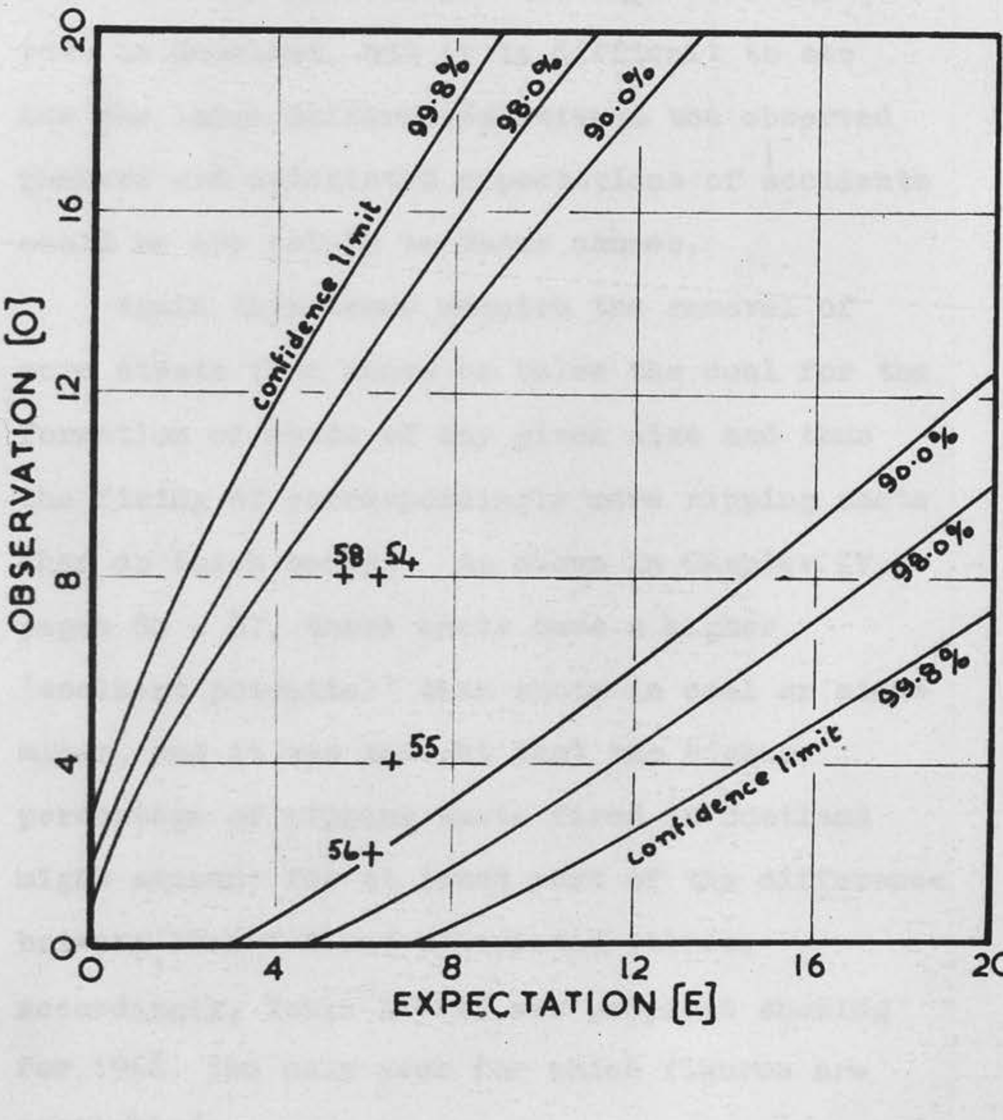
$$E = N \left(\frac{A-O}{T-N}\right)$$

Year	O	$N \times 10^6$	$T \times 10^6$	A	$E = N\left(\frac{A-O}{T-N}\right)$
1954	8	2.22	95.35	265	6.14
1955	4	2.28	95.67	272	6.55
1956	2	2.26	96.92	255	6.05
1957	8	2.18	95.56	249	5.60

GRAPH XXXII

SHOTFIRING ACCIDENTS IN THE CENTRAL EAST AREA, SCOTTISH DIVISION, 1954-1957.

The observed number of accidents [O] plotted against the expectation [E], calculated on the basis of the accident rate for the rest of the country



applies in seams lying at high inclinations where movement up or down the face may involve clambering from prop to prop rather than walking or crawling. The influence of these factors cannot be assessed with the information at present collected in accident statistics. Faulted ground will generally involve the firing of shots in the solid rock above or below the coal seam, and these shots included with ripping shots in 'other locations' were shown in Chapter IV to have a high potential risk. The combination of all these factors may provide a partial explanation for the high accident rate in Scotland, but it is difficult to see how the large differences between the observed numbers and calculated expectations of accidents could be due solely to these causes.

Again thin seams require the removal of more strata from above or below the coal for the formation of roads of any given size and thus the firing of correspondingly more ripping shots than do thick seams. As shown in Chapter IV, pages 84 - 87, these shots have a higher 'accident potential' than shots in coal or stone mines, and it was thought that the higher percentage of ripping shots fired in Scotland might account for at least part of the difference between observed and calculated values. Accordingly, Table XXVIII was prepared showing for 1952, the only year for which figures are available/

TABLE XXVIII

All Figures Relate to Year 1952

	Coal	Stone Mines	Other Locations	Total
Number of shots fired in Scottish Division (millions)	10.14	1.11	2.64	13.89
Numbers expressed as a percentage (P_1) of the total of 13.89 millions	73	8	19	100
Number of accidents in Scottish Division (N)	40	2	21	63
Accident rate/million shots in Scotland	3.94	1.80	7.95	
Numbers of shots, expressed as percentages of the total, for rest of country (P_2)	80	4.5	15.5	100
Calculated expectation $N \frac{P_2}{P_1}$	43.8	1.1	17.1	62.0

available, the distribution of shots and accidents and the corresponding accident rate for the three locations, 'coal', 'stone mines' and 'other locations'. The percentage distribution of shots is also shown for the rest of the country, and expectations for accident figures for Scotland worked out, using the accident rates calculated for Scotland and the distribution of shots for the rest of the country. It is seen that the overall difference between the observed number of accidents and the calculated expectation is very slight, the anticipated drop in accidents from ripping shots being almost balanced by a similar increase from coal shots.

The most obvious difference in shotfiring practice in Scotland, compared with the rest of Britain, is the continued use of considerable quantities of non-permitted explosive in conjunction with capped fuses at naked light collieries. Anyone firing shots where permitted explosives only may be used, must have the qualifications prescribed in Articles 4 and 5 of the Coal Mines (Explosives) Regulations, 1956, and these include, inter alia, a Certificate of Competency, a Shotfirer's Certificate granted on examination, or a Service Certificate. However, in naked light mines, a man may be appointed to be a shotfirer if he has certain practical mining experience and has attained the age/

age of 22 years, or 21 years and possesses a General Certificate in Mining or higher qualification. It might logically be expected that the employment of such unqualified men would lead to the occurrence of excessive numbers of shotfiring accidents.

In practice, it has been demonstrated both by the Royal Commission on Accidents in Mines (i) and by the Shotfiring and Its Alternatives Committee of the Institution of Mining Engineers (ii) that there is no substance in this argument. This result may be construed in two ways, and taken to indicate that fuse firing is no more dangerous than electric firing or that too much emphasis has been placed on the need for intensive training of shotfirers. It was shown on pages 54 - 56 of Chapter IV that the very high accident rate, which led to the formation of the committee of enquiry into the precautions necessary to secure safety in the use of explosives in coal mines, had begun to fall before the resultant legislation could be enacted and enforced. Unfortunately, this improvement has not been maintained and the most recent report of the Chief Inspector of Mines and Quarries reveals that a total of 294 people were killed and injured in shotfiring accidents in coal mines in 1958, an increase of 45 over the figure for 1957. It would thus appear that the amount of attention given to the training of shotfirers/

shotfirers is not yielding the expected benefits of accident reduction.

Having dealt with these factors without finding any reason for the high accident rate in Scotland, recourse must be made to less concrete but nevertheless possibly real explanations. The standard of practice achieved in the care in the use and handling of explosives offers a ready means of solution, as this could possibly override any or all other factors. The influence of this is very difficult to assess, as any visits to collieries by independent observers must be made with the approval and assistance of the National Coal Board and the purpose of the visit must, of course, be disclosed. Thus any section visited will be forewarned. In addition the observer will normally be accompanied by a senior official of the colliery, such as the Safety Officer, and the men will probably pay more attention to detail under this additional supervision than would normally be the case. To obtain worthwhile opinions, a very considerable period would have to be spent in several sections in each Division and this is not possible or practicable in a comparatively short period of study.

Again if any attempt is made to rely on the opinion of mining engineers with long and varied experience/

experience in different Areas or Divisions, another difficulty arises. Any person whose opinion might be expected to be valuable will normally have risen to hold a responsible post in the National Coal Board or the Mines Inspectorate and is thus no longer a private individual free to express opinions which may be quoted against him at some later date. In the author's experience such men either give opinions, later found by personal experience to have been wildly overrated, on the standard of practice achieved in their Area or Division, with permission to quote, or a more realistic appraisal with strict injunctions that the comments are not for publication. It must regrettably be concluded, therefore, that opinions on the standard of practice achieved would have to be gained personally in each Division and this would take an impracticable length of time. It is in fact doubtful if permission for such long visits, with the stated intention of obtaining a measure of the care employed in the use and handling of explosives, would be granted.

In the more restricted field of attempting to explain the high accident rate in Scotland, more positive opinions can be expressed. In the course of these investigations, and other visits unconcerned with shotfiring, a total of six weeks were spent in the East Midlands Division/

Division, and approximately nine months in Scottish pits. Nothing was seen which could suggest that the standard of practice in shotfiring was lower in the Scottish Division than in the East Midlands and there was no reason to suppose that the high accident rate in Scotland was due to this factor. The impression was gained, however, that the shotfirer did not command the same degree of respect in Scotland. While this is unlikely to have any immediate effects, it must be remembered that shotfiring accidents occur very infrequently, and individual local factors need operate only very occasionally to result in accident rate variations.

It has been suggested, that the Scot, notoriously difficult to discipline and always ready to take a chance, might be more prone to accidents on this account. Certainly these characteristics, desirable in certain circumstances, are very much out of place in shotfiring operations. It is difficult to see why this should apply only to shotfiring as Wynn (iii) showed that the Scottish Division was significantly better than the rest of the country in accidents from falls of ground at the face. Approximately the same people are involved and why shotfiring should be characterised by foolhardiness, and roof control by higher than average standards of care is difficult to understand. Some explanation of this may be found/

found in the following figures, abstracted from the 1958 Annual Report of the Chief Inspector of Mines and Quarries, illustrating the order of the difference between the risks incurred from shotfiring operations and falls of ground at the face:

	Nos. Killed	Nos. Seriously Injured
Falls of ground at the face	56	237
Shotfiring	6	38

In these figures 'seriously injured' refers to those people whose injuries would be reportable to the Mine Inspectorate by reason of their severity, rather than the nature of their occurrence.

Conclusions

1. It is seen from the accident figures recorded for the North Eastern Division, and the Central East Area of the Scottish Division, that off-shift firing confirms its obvious theoretical advantages with practical reductions in the shotfiring accident rate.

2. The provision of portable shelters does not result in any reduction in accidents.

3. It has not been found possible to draw any positive conclusions for the high accident rate in Scotland, and certain factors which might be expected to raise the incidence of accidents are shown to have a small or negligible effect. It is/

is suggested that the combination of difficult geological conditions, and a national temperament notoriously resistant to discipline may result in the high figures shown.

REFERENCES

(i) Report of the Royal Commission on Accidents in Mines, 1938

(ii) Third Report of the Institution of Mining Engineers Shotfiring and its Alternatives Committee. Transactions of the Institution of Mining Engineers, CXI, 545.

(iii) Wynn. Applications of Probability Theory to the Study of Mining Accidents. Safety in Mines Research Establishment, Research Report, No. 7

CHAPTER X

CONCLUSIONS AND RECOMMENDATIONS

Conclusions

1. Although the hazard involved in the use and handling of explosives, expressed as persons killed or injured per million shots fired, is not high, the large number of shots fired annually results in a comparatively large number of accidents.

2. Shotfiring accidents may be divided into two main categories - those due to some failure of the explosive or ancillary equipment to attain perfection and those due to failure of the human element. Continued research and develop:ment in explosives technology have resulted in a reduction in the former class but the latter type forms a large and increasing percentage of the annual total of accidents.

3. The statistical techniques developed by Wynn and used in this work provide a valuable means of representing graphically the significance of any differences existing between calculated expectations and observed numbers of accidents.

4. Only two National Coal Board Divisions showed significant differences in accident rates from the rest of the country over the period for which figures were available. The Scottish Division was very much worse, and the North Eastern Division very much better than the average for the rest of the country.

On the limited amount of information available, it appears that shots in locations other/

other than coal seams or stone mines present a hazard greater than would be expected from the relative numbers fired in these situations.

5. Over half the annual total of shots fired in National Coal Board collieries are in coal, initiated by electric detonators and fired singly, and this class is still by far the most important in British coal mining practice. Simultaneous and delay firing offer potential safety as well as operational advantages, but no measure of these can be obtained with the limited information at present available.

6. Alternatives to explosives offer safety advantages over conventional explosives and should give corresponding benefits in a reduction in accidents.

7. Pulsed infusion shotfiring, in the comparatively limited circumstances in which it may be usefully employed, yields safety, operational and economic advantages over conventional shotfiring.

8. Limited evidence suggests that the use of water ampoules in conjunction with conventional explosives gives consistent reductions in the dust produced by shotfiring in rippings but widely variable figures have been recorded in the dust counts before and after the introduction of these devices in coal blasting operations.

9./

9. Off-shift firing offers great potential advantages in comparison with conventional practice, and these are confirmed by the low accident rate in the North Eastern Division and the Central East Area of the Scottish Division.

10. The general standard of practice achieved in shotfiring is very much below the minimum required by law and demonstrated to the shotfirer in the training period. This is due to a combination of lack of effective supervision and the needs of production clashing with safety requirements.

11. In orthodox hand-filled longwall working when the coal preparation shots are fired on the production shift, the nature of the shotfirers work, with periods of great activity alternating with periods of comparative idleness, leads to carelessness and skimped precautions.

12. In orthodox longwall working, very limited shelter is available on the coal face, and it is frequently necessary to rely on cover which cannot possibly be considered satisfactory.

13. It is seen from the accident positions of the North Eastern, North Western and East Midlands Divisions that the provision of portable shelters in gate roads effects no sensible reduction in shotfiring accidents.

14./

14. Although no satisfactory concrete explanation for the high shotfiring accident rate in the Scottish Division could be found, it is suggested that a combination of difficult natural conditions, making men more reluctant to retreat a sufficient distance from the shots and a national temperament not amenable to discipline might explain the difference between Scotland and the rest of the country.

Recommendations

1. All shotfiring accident reports should include the location and method of initiation of the shot causing the accident. When sufficient information has been collected, the relative hazards involved in the firing of shots by different methods in different locations could be accurately assessed, and any changes in practice, indicated by the results to be desirable, could be effected.

2. Whenever possible shots should be fired when the least possible number of men are exposed to the risk of flying debris. When the introduction of off-shift firing would create roof control difficulties it might be possible to fire a limited number of shots on the preparation shift and the remainder on the production shift.

3. As there is no indication that the provision of shelters in gate roads results in a reduction in/

in shotfiring accidents, emphasis should be placed on the necessity for making and using the solid cover provided by manholes. These should be made even if not required by law.

If care were taken in these steps, accidents from shots in rippings and stone mines could be virtually eliminated.

4. As little shelter exists on a longwall face, it is difficult to see how accidents resulting from coal preparation shots will ever be completely eliminated. However, reductions might be effected in two ways:

(a) At pit level, attempts should be made to arrange the method of work in the sections to reduce the need for high speed, and correspondingly reduced care, in shotfiring operations.

(b) Although material may be projected from any shot, the possibility is increased should the shothole be badly placed, overcharged or understemmed, and these are all factors militating against the economic, as well as safety needs of shotfiring. As experience has shown that the improved results following an increased supervision tend to be lost when the supervision is removed, it is suggested that the appointment of a specialist oversman to supervise blasting practice would reduce these malpractices, and bring about attendant safety, as well as economic, advantages.

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